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In Situ Mining Research

Proceedings: Bureau of Mines Technology Transfer Seminar, Denver, Colo., August 5, 1981

Compiled by Staff–Bureau of Mines

UNITED STATES DEPARTMENT OF THE INTERIOR

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UNITED STATES DEPARTMENT OF THE INTERIOR
James G. Watt, Secretary

BUREAU OF MINES

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PREFACE

This Information Circular summarizes recent Bureau of Mines results covering in situ mining research. The papers are only a sample of the Bureau's total effort to improve minerals productivity through its Resources Technology Program, but they represent the major research effort in the in situ mining area. Those desiring more information on the Bureau's Mineral Resources Technology Program in general, or information on specific research, should feel free to contact the Bureau of Mines, Division of Mineral Resources Technology, 2401 E Street, N. W., Washington, D.C. 20241, or the appropriate author listed in the following proceedings.

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IN SITU MINING RESEARCH

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Denver, Colo., August 5, 1981

Compiled by Staff—Bureau of Mines

ABSTRACT

These proceedings consist of an overview of the in situ mining research currently being carried out by the Bureau of Mines. The following papers emphasize two general aspects of the in situ mining method: the environment and productivity. Both areas are extremely important, particularly because in situ leach mining is a relatively new mining method from a commercial point of view. Topics covered include the restoration of ground water, the selection of lixiviants, in situ mining of commodities other than uranium, in situ mining costs, the application of resistance measurements to in situ mining, an acid leach mining case history, and the use of branched boreholes for in situ mining. A bibliography of Bureau of Mines publications on in situ mining is appended.

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INTRODUCTION

by

Dennis V. D'Andrea[1]

In situ leach mining is a relatively new method that has the potential of recovering a variety of mineral commodities such as copper, uranium, gold, silver, manganese, and nickel. This mining method can be applied to smaller or lower grade deposits that would otherwise not be mined, and also has major advantages when compared with conventional mining in the areas of health and safety and environment. Past experience has indicated that lower capital costs are required for in situ mining, and there is a quicker return on investment.

Copper and uranium have been the two primary commodities extracted by in situ mining. In situ leaching of copper oxide deposits has been carried out at five locations in the Southwest. During 1980 there were 16 commercial-scale in situ uranium leaching operations at various stages of production and construction which accounted for about 10 percent of the domestic uranium production. Numerous companies have recently expressed interest in in situ mining other commodities such as manganese, gold, and silver, but there are presently no commercial operations.

The Bureau of Mines began conducting research in 1971 to develop improved in situ leach mining techniques and to minimize environmental risks. The appendix lists publications that describe the in situ mining research that has been conducted or coordinated by the Bureau. Major research areas investigated include well construction techniques, computer simulation, reducing environmental concerns, borehole mining, blasting to increase permeability, and economic analyses. The initial research was directed toward oxide copper deposits. In 1975 the emphasis shifted toward uranium in situ leaching mining, and current research is aimed at development of in situ mining methods for the recovery of a variety of mineral commodities.

The goal of the Bureau's in situ leach mining investigations is to accelerate the development and transfer to industry of improved techniques for in situ mining of marginal deposits, thus expanding the Nation's supply of critical mineral commodities.

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IN SITU LEACH MINING--
CURRENT OPERATIONS AND PRODUCTION STATISTICS

by

William C. Larson[1]

ABSTRACT

Thus number of in situ leach operations has increased significantly since 1975. As of May 1980, there were 27 active projects, including 18 commercial-scale operations (some of them under construction) and 9 pilot scale operations.

The south Texas uranium district and Wyoming have been the most prominent areas in early field experiments as well as in commercial applications of this new recovery technique, and in situ leach tests are now being conducted in Colorado and New Mexico. The growing number of commercial-scale operations is evidence that in situ mining now offers a third option along with open pit and underground mining for winning uranium from sandstone host rocks. It is estimated that in 1979 about 9 percent of the Nation's total uranium production was from in situ mining.

INTRODUCTION

In situ leach mining should no longer be considered a "last resort" method for recovery of uranium in sandstone host rocks. More and more operators are turning to this recovery technique as a viable alternative to conventional open pit or underground methods. By way of definition, in situ leach mining is that method where the ore mineral(s), in the original geologic setting, is preferentially leached from the host rock by the use of specific leach solutions and the mineral value(s) recovered. Briefly, in situ uranium mining consists of (1) injecting a suitable leach solution into the ore zone below the water table, (2) oxidizing, complexing, and mobilizing the uranium, and (3) recovering the pregnant solution through production wells for processing through an ion exchange system to recover the uranium.

As is typical in the development of any new technology, there has been little public information available on in situ leach mining, particularly before 1977. In the past few years, however, in situ leach mining has evolved from totally experimental to commercial status. Thus, a number of papers have been published in the past 2 or 3 years on subjects related to in situ mining, such as design criteria, operating procedures, costs, environmental information, and general state-of-the-art information.

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[1] Supervisory mining engineer, Twin Cities Research Center, Bureau of Mines, Minneapolis, Minn.

One aspect of in situ uranium leach mining that has not been discussed in the literature to any great extent is production data. This type of information is of great importance to the operators, to manufacturers associated with in situ leaching supplies, and to mineral forecasters, bankers, and individuals who make decisions regarding uranium exploration. The past 5 years have seen a significant growth in the in situ uranium mining industry, particularly in Texas and Wyoming, and many people may not be aware of the impact that in situ uranium mining has had on this country's uranium production, or of its potential for future production. The following discussion centers on two areas of in situ uranium mining: current operations and production statistics.

CURRENT OPERATIONS

Since the early 1960's research and development efforts have yielded significant advances in in situ uranium mining technology. The first modern in situ uranium leach mine was operated by Utah Construction and Mining Co., now Utah International Inc., at its Shirley Basin site in Wyoming. Utah Construction used many of the same principles and techniques that are currently in use, such as continuous ion exchange systems, pattern drilling, and the use of leach solutions with an oxidizer. During 1961-63, the company experimented with many techniques, particularly with regard to well development procedures and leach solutions. By 1963, the company had experimented with and tried 5 generations of well field designs and had drilled over 100 well field patterns in an attempt to optimize recoveries. From 1963 to 1969 in situ mining was the only method used by this company for uranium production. After 1969 the in situ leach operation was replaced by open pit mining.

Between 1969 and the early 1970's numerous research and development activities were taking place in the industry, and pilot tests expanded from the laboratory into the field. In the mid-1970's small-scale pilot tests were being conducted in Wyoming, New Mexico, and Texas.

Following successful field testing at the Clay West site, the Atlantic Richfield Co. initiated the first commercial-scale in situ uranium mining operation in Texas in 1975. The Clay West mine, operated by U.S. Steel Corp., is located in Live Oak County northwest of Corpus Christi. Twelve additional commercial-size operations in Texas have been in various stages of production since the startup of the Clay West site. Table 1 summarizes the status of in situ uranium leach mining operations in Texas.

TABLE 1. - Status of uranium in situ leach mining operations in May 1980

Firm	Operation	
	Commercial scale	Pilot scale[1]
TEXAS		
Caithness	-	McBryde.
Conoco	-	Trevino.
Everest Minerals Corp	Hobson	-
Intercontinental Energy Corp	Zamzow, Pawnee[2]	-
Mobil Oil Co	Holiday-El Mesquite, Nell, O'Hern.	-
Texaco, Inc	-	Hobson,
Union Carbide Corp	Palangana	-
Uranium Resources Inc	Benavides, Longoria	-
U.S. Steel	Burns	-
U.S. Steel--N.M.U. Inc	Boots, Clay West, Moser	-
Wyoming Mineral Corp	Bruni, Sulfur Creek	-
WYOMING		
Cleveland Cliffs joint venture	-	Collins Draw.
Exxon Minerals U.S.A.	Highland[3]	-
Kerr-McGee	-	Bill Smith.
Nubeth joint venture	-	Sundance.
Ogle Petroleum	Bison Basin[3]	-
Rocky Mountain Energy	-	Reno Ranch.
Rocky Mountain Energy joint venture	Nine Mile Lake[4]	-
Teton Exploration	-	Luemberger.
Wyoming Mineral Corp	Irigaray	-
NEW MEXICO		
Mobil Oil Co	-	Crown Point
COLORADO		
Union Oil-Power Resources joint venture	Keota[5]	-

[1] Pilot-scale operations other than those listed are known to exist in Texas, Wyoming, and New Mexico.
[2] Restoration stage.
[3] Commercial scale planned 1980.
[4] Commercial scale planned 1983.
[5] Commercial scale planned 1981.

Encouraged by the apparent success of in situ uranium leach mining, companies have pilot tests in operation or under construction in other major uranium-producing States, such as Wyoming, New Mexico, and Colorado. As of May 1980, there were eight operators in Wyoming in some stage of in situ leaching development, covering nine projects. Several operators have completed research and development tests at more than one site. Table 1 shows the status of in situ uranium leaching mining operations in Wyoming as of May 1980. Finally, Colorado and New Mexico each had at least one active pilot-scale operation as of May 1980, as shown in table 1. These figures bring the total number of projects in all states to 27, including pilot or commercial scale.

Several other States have received increased interest in in situ uranium leaching, including Montana, Arizona, South Dakota, and California, although as yet no pilot-scale studies have been initiated in these States. The above examples were obtained from a variety of public sources of information, and undoubtedly other in situ uranium mining projects are in various stages of planning.

PRODUCTION STATISTICS

The previous section discussed the growth of in situ uranium mining operations through 1980. This section discusses the in situ uranium mining production capabilities and estimated production from 1975 through 1982. There are two reasons for presenting the following material. First, very little information has been published in the literature, and therefore a void exists in this area. Second, uranium production from in situ mining is a new technology, and many people may have underestimated its growth during the past 5 years. For example, table 2 shows the published figures on rated capacities of the commercial-scale in situ mining operations. Companies often publish, in a variety of sources, a figure that represents the rated annual capacity of an operation, given suitable head grades to the processing plant as well as anticipated flow rates. Such production figures are realistic based on the information available at the time the plant was built. They are not actual uranium in situ mining production figures, but they do give a base or frame of reference from which to estimate actual production.

TABLE 2. - Rated capacities of commercial-scale uranium in situ leach mining operations--current and near-term projects

Operation	Rated capacity	
	Lb/yr	Kg/yr
Everest Minerals Corp. (Hobson)	150,000	68,000
Exxon Minerals--U.S.A. (Highland)	750,000	340,200
Intercontinental Energy Corp. (Zamzow)	250,000	113,400
Mobil Oil Co. (Holiday-El Mesquite)	650,000	294,800
Mobil Oil Co. (Nell)	100,000	45,300
Mobil Oil Co. (O'Hern)	175,000	79,400
Ogle Petroleum (Bison Basin)	400,000	181,400
Rocky Mountain Energy joint venture (Nine Mile Lake)	500,000	226,800
Union Carbide Corp. (Palangana)	300,000	136,000
Union Oil--Power Resources joint venture (Keota)	500,000	226,800
Uranium Resources Inc. (Benavides)	300,000	136,000
Uranium Resources Inc. (Longoria)	100,000	45,300
U.S. Steel (Burns)	1,000,000	453,600
U.S. Steel--N.M.U. (Boots, Clay West, Moser)	1,000,000	453,600
Wyoming Mineral Corp. (Bruni)	250,000	113,400
Wyoming Mineral Corp. (Irigaray)	500,000	226,800
Wyoming Mineral Corp. (Sulfur Creek)	500,000	226,800

In the last 5 years, production from in situ mining has increased considerably. Table 3 shows an historical comparison between the growth rates (rated capacities) of the industry and the estimated uranium production. If this trend continues, in situ uranium mining will be a logical third alternative for the extraction of uranium from sandstone host rocks. Table 4 shows the percentage of uranium produced by in situ mining compared with the annual production of uranium by all methods. As can be seen from this figure, uranium production by in situ mining, particularly since 1976, has been significant.

TABLE 3. - Estimated production versus rated capacities of uranium in situ leach mining operations, 1975-82

Year	Rated capacity		Estimated production	
	Lb/yr	Kg/yr	Lb/yr	Kg/yr
1975..................	500,000	226,800	150,000	68,000
1976..................	1,000,000	453,600	500,000	226,800
1977..................	2,700,000	1,224,700	1,300,000	589,600
1978..................	2,975,000	1,349,400	2,200,000	997,900
1979..................	3,700,000	1,678,000	3,000,000	1,587,600
1980[e].................	5,700,000	2,585,500	4,000,000	1,814,300
1981[e].................	6,500,000	3,084,400	4,300,000	2,086,500
1982[e].................	7,650,000	3,583,400	5,600,000	2,540,000

[e] Estimated.

TABLE 4. - Percentage of uranium produced by in situ mining compared with production by other methods

Year	Total concentrate production,[1] lb	In situ mining productivity,[e] lb	Percent of total
1975............	23,200,000	150,000	0.6
1976............	25,494,000	500,000	2
1977............	29,880,000	1,300,000	4
1978............	36,980,000	2,200,000	6
1979............	37,460,000	3,000,000	8
1980[e]...........	40,000,000	4,000,000	10

[e] Estimated.

[1] Statistical Data of the Uranium Industry. Dept. of Energy, 650-100(80), Grand Junction, Colo., 1980.

SUMMARY

In summary, several observations can be made regarding uranium in situ mining in the United States. First, the number of in situ mining operations in the United States is expanding at an impressive rate. Second, the estimated production figures show that uranium produced by in situ mining is a significant percentage of the Nation's uranium output. Third, in the opinion of many, this mining method is a viable alternative for the recovery of uranium from sandstone host rocks.

GOLD AND SILVER LEACHING PRACTICES IN THE UNITED STATES

by

Peter G. Chamberlain[1] and Michael G. Pojar[2]

ABSTRACT

Many new gold and silver mining operations have been established as a result of higher gold and silver prices. Leaching processes capable of extracting gold and/or silver from small deposits and/or lower grade ores have become attractive to many precious metal mine operators. This paper discusses operating principles associated with gold and silver leach mining. Problems confronting potential leaching operations are also discussed along with research projects in progress to resolve these problems.

INTRODUCTION

Treatment methods applicable to comparatively high-grade gold and silver ores include gravity concentration, amalgamation, flotation, cyanidation, or direct smelting. Such processes involve high capital investments as well as high operating costs. A conventional cyanidation plant used in processing gold and silver ores usually includes crushing, fine grinding, and agitation leaching in cyanide solutions, countercurrent decantation in thickeners for separating the pregnant solution, clarification of this solution by filtering, deaeration by vacuuming, and precipitation of the precious metals by zinc powder. It is obvious that such a processing scheme is costly from the viewpoint of capital investment and operating cost. For this reason, such processes are not economically justified for lower grade ores.

Many of the known and newly discovered gold and silver deposits are low in gold and/or silver content, have limited reserves, or contain other minerals that make processing by conventional gravity and cyanidation methods impractical. Such lower grade and refractory deposits pose a big challenge to modern extraction technology.

The gold and/or silver in small and low-grade deposits for which conventional mining and milling are too costly might be economically recoverable by leaching or solution mining methods. Solution mining is the extraction of metals by leaching from ores located within the confines of a mine, or in dumps, ore heaps, slag piles, and tailing ponds.

If the ore is mined or gathered from old mine waste rock piles and hauled to specifically prepared pads for leaching, the method is termed "heap" leaching (fig. 1). The rock is frequently, but not always, crushed before being

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[2] Mining engineer, Blasting Technology and In Situ Mining.
Both authors are with the Twin Cities Research Center, Bureau of Mines, Minneapolis, Minn.

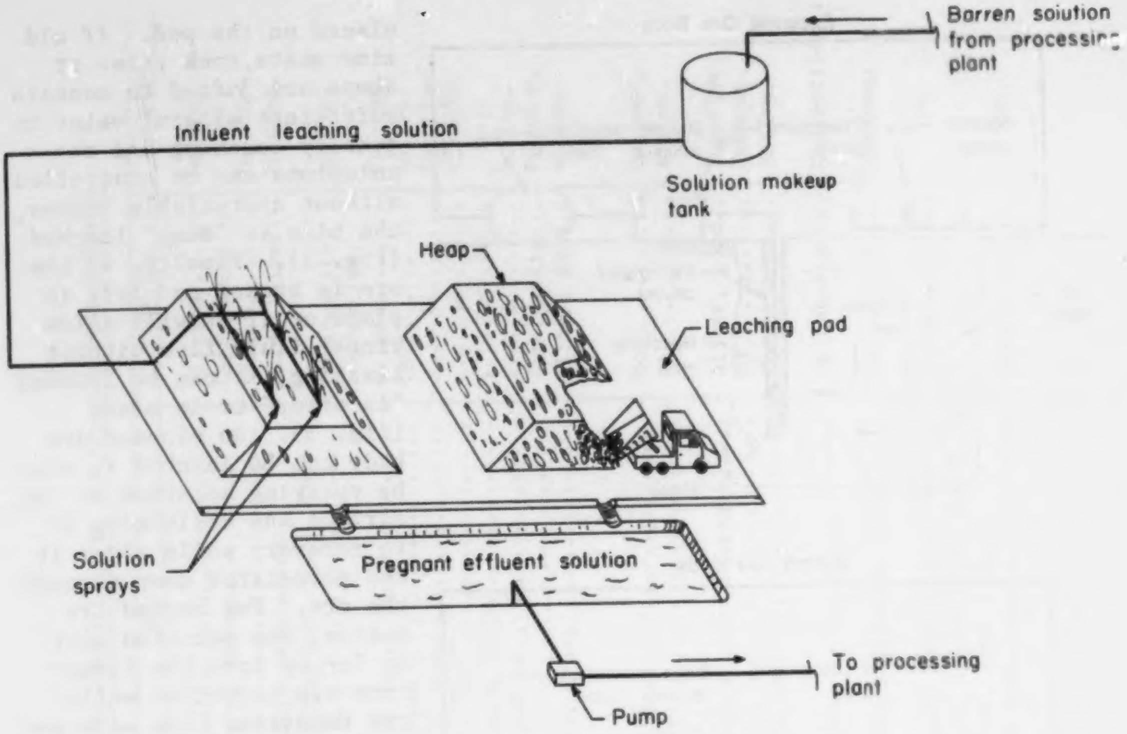
Barren solution from processing plant
Influent leaching solution
Solution makeup tank
Heap
Leaching pad
Solution sprays
Pregnant effluent solution
To processing plant
Pump

FIGURE 1. - Schematic of a heap leaching operation.

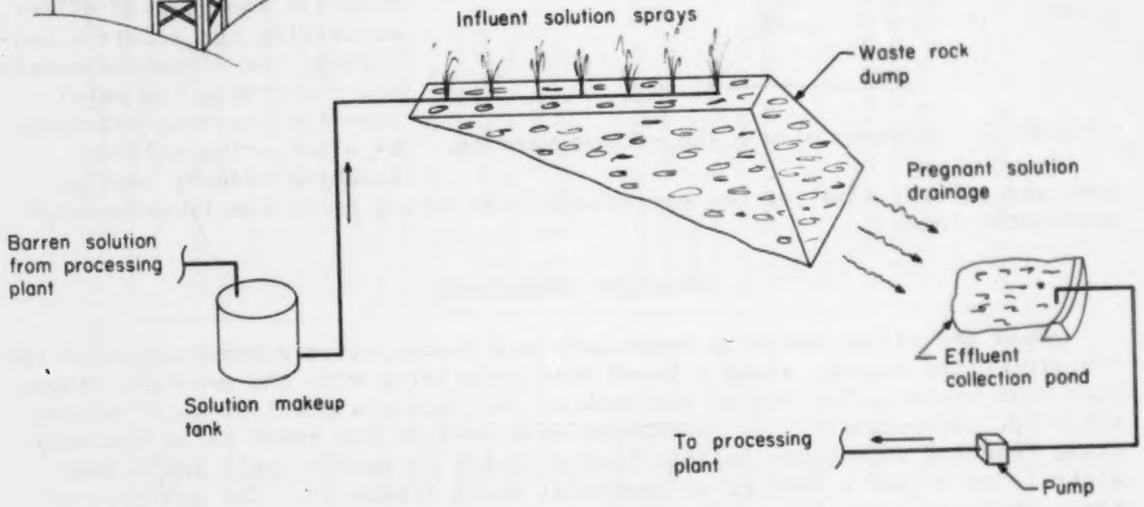
Influent solution sprays
Waste rock dump
Pregnant solution drainage
Barren solution from processing plant
Effluent collection pond
Solution makeup tank
To processing plant
Pump

FIGURE 2. - Schematic of a dump leaching operation.

placed on the pad. If old mine waste rock piles or dumps are judged to contain sufficient mineral value to justify leaching and the solutions can be controlled without appreciable losses, the pile is "dump" leached (fig. 2). Finally, if the ore is broken and left in place or if it will allow proper fluid flow without blasting, it can be leached "in situ," or in place (fig. 3). An exposed ore body can be leached in situ by spraying solution on the surface and collecting it in recovery wells after it has percolated down through the ore. For buried ore bodies, the solution must be forced into the formation via injection wells and recovered from adjacent recovery wells.

Basically leaching involves spraying a cyanide solution onto the ore, or the injection of a cyanide solution into an ore body to dissolve the gold or silver, collecting the solution containing the dissolved metals and recovering the metal from the leaching solution. By eliminating milling, leaching reduces capital cost and startup time for new operations. Operating costs are likewise significantly lower.

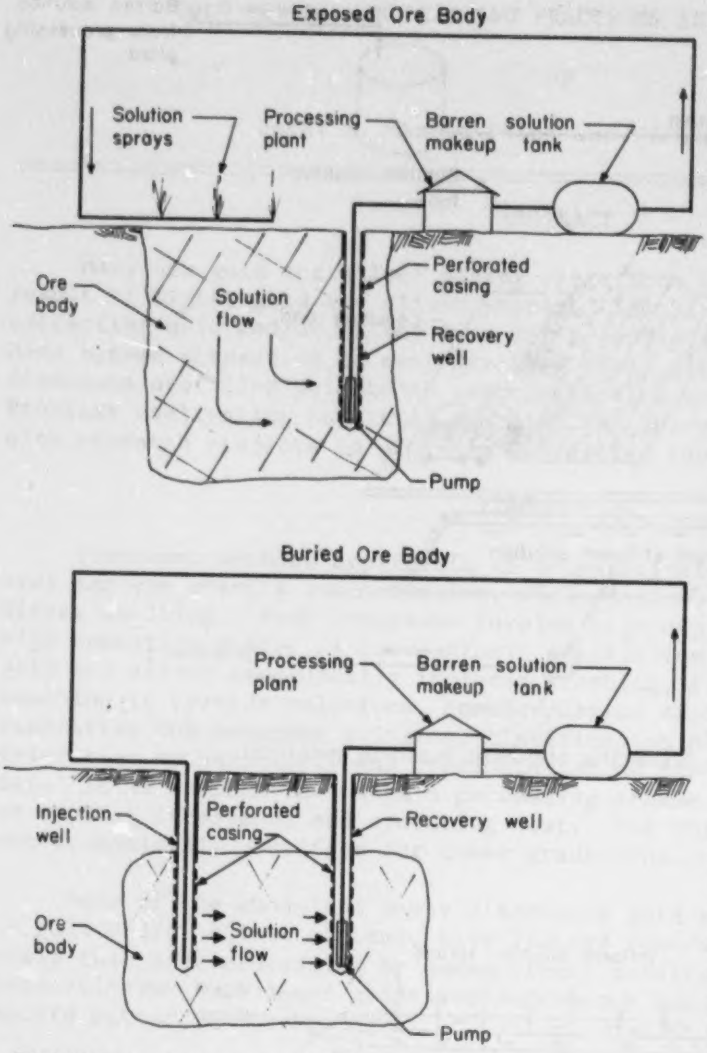
Exposed Ore Body
Solution sprays
Processing plant
Barren makeup
solution tank
Ore body
Solution flow
Perforated casing
Recovery well
Pump
Buried Ore Body
Processing plant
Barren makeup
solution tank
Injection well
Perforated casing
Recovery well
Ore body
Solution flow
Pump

FIGURE 3. - Schematic of an in situ leaching operation.

LEACHING OPERATIONS

Gold and silver leaching operations are concentrated predominantly in the Western United States, along a broad belt coinciding with the mountain ranges that have historically hosted the bulk of our Nation's precious metal mining activity. Approximately 84 operations have been or are known to be actively using leaching techniques (either heap or dump) to extract gold and silver minerals on either a test or a commercial scale (table 1). The majority of these operations are located in Nevada and Arizona. The principal ones are shown on figure 4.

TABLE 1. - Gold and silver heap and dump leaching operations in the Western United States

State and county	Company	Mine	Mineral	Status
Arizona:				
Cochise	Tombstone Exploration, Inc.	Contention	Silver	Active.
Do	State of Maine Mining	State of Maine	Cerargyrite	Do.
Do	71 Minerals (Sierra Minerals)	NA	Silver	Inactive.
Do	Silver Ridge Mining--Houston Mining and Resources.	Nicholas, Gambasinos, Rattling Boy, Stuck Steel.	Gold, silver	Planned.
Pinal	Vekol Mine Development--Sunburst Mining Co.	Vekol	Cerargyrite, bromyrite.	Active.
Yavapai	New Jersey Zinc	Silver Clip, Black Rock	Silver	Planned.
Do	Congress Consolidated Gold	Congress	Gold	Active.
Do	Walter Statler	Little Jessie	do	Inactive.
Yuma	Dr. Eugene Burdick	North Star	do	Planned.
Do	Magini Leasing and Contracting	Robinson Claims	do	Do.
Do	Hildebrand Drilling	San Marcos	do	Active.
Do	Red Cloud Mining	Red Cloud	Cerargyrite	Planned.
NA	AMCA Industries, Ltd	Silver Cross	Silver, gold	Active.
Colorado:				
Gilpin	Nuclear and Minerals Corp	Leaching site	Gold	Unknown.
Mineral	Minerals Engineering--Chevron Resources Co.	Emperius	NA	Active.
Do	Minerals Engineering	Cowboy Johnson (Corsair)	Silver, gold	Inactive.
Teller	Gold Hills Mesa Corp	NA	NA	Unknown.
Do	Gold Resources--Newport Minerals Inc.	Globe Hill	Gold	Active.
Do	Golden Cycle Corp.--Texasgulf Inc.	Ajax	Calaverite	Conventional.
Do	Merchant-Caithness joint venture--Venture Mining.	Stratton Estates	NA	Unknown.
Do	National Energy Corp	Midget, Moon Anchor, Red Bird, Yellow Bird, Dolly V, Atlas, Loan Jack.	Silver, gold	Do.
Do	Gold Ray Mining	Gold Ray	Gold	Do.

NA Not available.

TABLE 1. - Gold and silver heap and dump leaching operations in the Western United States--Continued

State and county	Company	Mine	Mineral	Status
Idaho:				
Custer	NA	NA	Gold	Planned.
Valley	Canadian Superior Mining--Ranchers Exploration and Development.	Yellow Pine (West End, Garnet Creek).	Gold, antimony	Do.
Do	Thunder Mountain Gold Inc.--Canadian Superior Mining.	Sunnyside Mine and surrounding claims.	Gold	Do.
Montana:				
Broadwater	United Minerals	NA	do	Do.
Jefferson	Lacana--Falcon Exploration	Tourmaline Queen	NA	Active.
Do	Placer Amex	Golden Sunlight	Gold	Inactive.
Phillips	Pickle Crow Exploration	NA	Silver, gold	Unknown.
Do	Zortman Mining Co	Ruby Gulch	Gold, silver	Active.
Do	Landusky Mining Co	August, Gold Bug	do	Do.
Powell	Adams Bros. Construction	Viking	Gold	Planned.
Nevada:				
Churchill	Desert Star Mining	Desert Star	Gold, silver	Active.
Do	Fisk and Son	Gold Hill	Gold	Do.
Clark	Crescent Mining, Ltd	Rest	do	Do.
Do	Big Delta Refinery, Inc	Dawn Renae	do	Do.
Do	Intermountain Exploration Co	Intermountain Limited Partners.	Gold, silver	Unknown.
Elko	Tuscarora Associates	Tuscarora Associates Plant	do	Active.
Do	McNeely Mining Contractors, Inc.	do	do	Do.
Esmeralda	Diablo Mine Services Co	Goldfield Ltd	Gold	Do.
Do	Mid-Continent Mining--Sunshine Mines.	Nivloc Sixteen to One Properties.	NA	Inactive.
Do	Falcon Exploration	Tonopah-Divide	Gold	Planned.
Eureka	Windfall Venture	Windfall	do	Active.
Do	Carlin Gold Mining Co	Carlin, Bootstrap	Gold, mercury.	Do.
Do	Newmont Mining Co	Maggie Creek	Gold	Planned.
Do	do	Gold Quarry Prospect	do	Do.
Humboldt	Bauer Metals, Inc	Martin Creek	Silver	Active.
Do	Lion Mines, Ltd	NA	Gold, silver	Planned.
Do	Lacana--Rayrock	Pinson Gold	Gold	Do.
NA	American Pyramid Resources	NA	do	Unknown.
NA	Dallas Exploration	NA	Silver, gold	Do.
NA	El Plata Mine	Leopard, Cornucopia	Gold, silver	Do.

NA	Volcanic Gold, Inc	NA	Gold	Planned.
NA	Intermountain Exploration Co	Intermountain Limited Partners.	Gold, silver	Unknown.
NA	Aaminex Gold	NA	Gold	Do.
Lander	Placer Amex--Bunker Hill	Cortez, Gold Acres	do	Active.
Do	New Pass Resources, Inc	New Pass Mill	Gold, silver	Do.
Do	Aaron Mining, Inc	Gold Quartz	do	Do.
Do	Duval Corporation	Copper Canyon	do	Do.
Mineral	Occidental Minerals Corp.--Candelaria Partners.	Candelaria project	do	Do.
Do	Hugh C. Ingle, Jr	Ashby Mine and Mill	Gold	Do.
Do	Ladd Enterprises	Ladd Tungsten Claims	Silver, gold	Unknown.
Nye	Smoky Valley Mining Co.--Copper Range.	Round Mountain	Gold	Active.
Nye	Ibex Mining Corp	Keystone	Gold, silver	Inactive.
Nye	Golden Arrow, Inc	Golden Arrow	do	Active.
Nye	Cyprus Exploration	Northumberland	Gold	Planned.
Nye	Summa Corp	NA	do	Inactive.
Pershing	Buckeye Mining Enterprises	Standard Gold	do	Active.
Do	Ore-Nugget, Ltd	Florida Canyon Gold Deposit	do	Do.
Do	Flying J Mines	Florida Canyon Mine and Mill.	do	Do.
Do	D Z Exploration Co	Packard	Silver	Do.
Do	Inland Resources	Twin Buttes	Gold	Do.
Storey	Intermountain Exploration Co	Comstock Lode	Gold, silver	Unknown.
Do	Flowery Gold	Flowery, Lady Bryan	Gold	Do.
Do	Minerals Engineering Co	Con-Imperial, Con-Chollar, Dayton.	NA	Do.
White Pine	Gold Creek Corp.--Diamond Silverado Exploration.	Diamond Silverado	Silver	Active.
Do	Gold Creek Corp	Treasure Hill	Lead, silver	Do.
Do	American Selico--Occidental Minerals Corp.	NA	Gold	Planned.
New Mexico:				
Catron	Challenge Mining Co	Eberle, Confidence	Gold, silver	Active
Santa Fe	Gold Fields Mining Co	Ortiz Project	Gold	Do.
NA	Canorex Development	NA	do	Inactive.
South Dakota:				
Lawrence	Cyprus Exploration	Gilt Edge	do	Planned.
Do	Tiaga Gold Corp--Wharf Resources	Anne Creek	do	Do.

NA Not available.

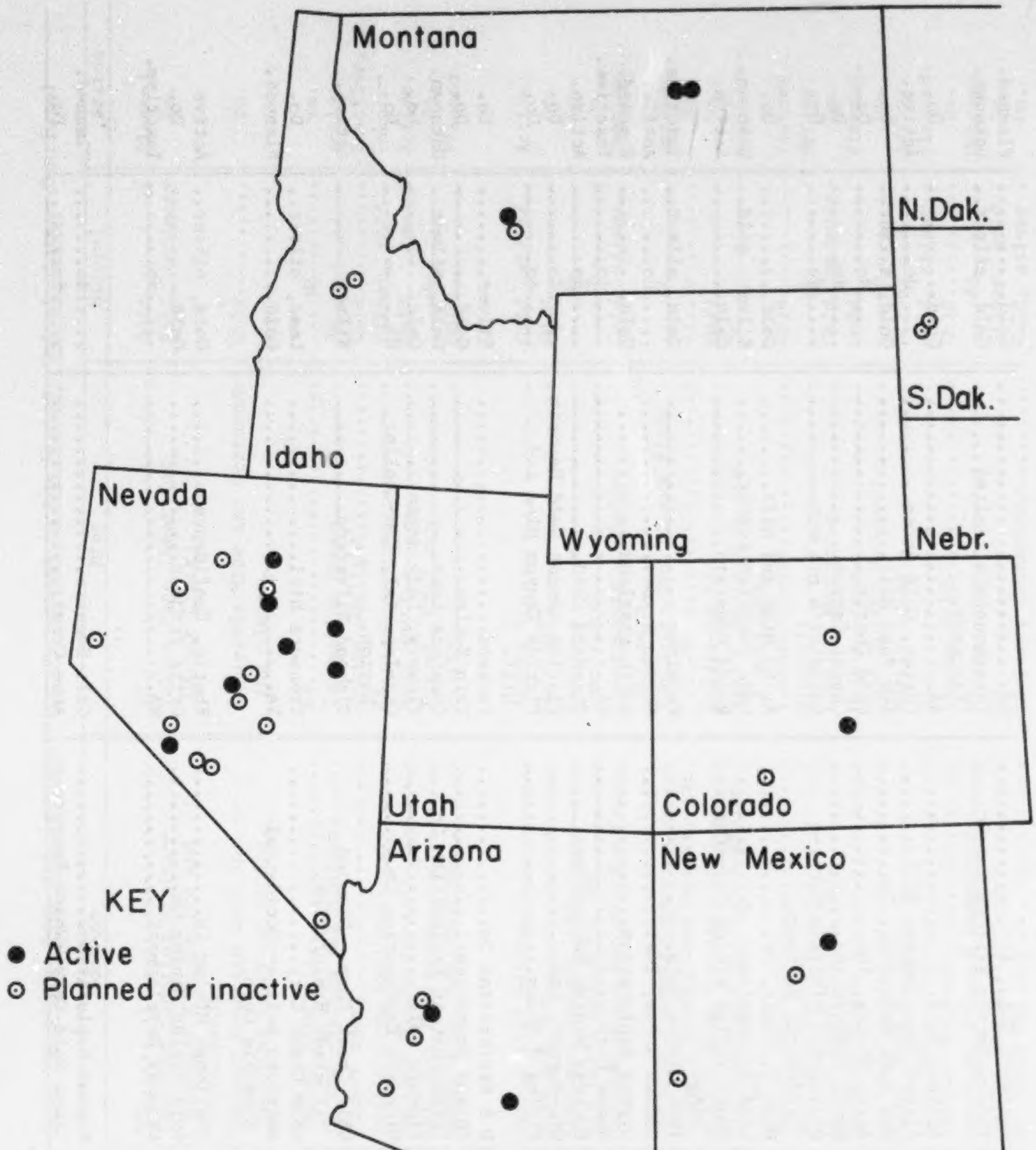
Montana
N. Dak.
S. Dak.
Idaho
Nevada
Wyoming
Nebr.
Utah
Colorado
Arizona
New Mexico
KEY
● Active
⊙ Planned or inactive

FIGURE 4. - Gold and silver leaching operation location.

The extractable gold is generally deposited as native or free gold, often associated with pyrite. Silver is generally deposited in compound form. The easiest ores to leach are those that have been weathered or oxidized. The average or typical ore grades that can be successfully leached economically include gold ores ranging from 0.01 to 0.03 oz Au/ton, and silver ores ranging from 1.0 to 4.0 oz Ag/ton.

Ores that are treated by heap leaching are--

1. Mined or gathered together from lean ore or waste dumps.
2. Crushed (optional).
3. Placed on specially prepared, lined leach pads using scrapers, trucks, or bulldozers.
4. Leached with sodium cyanide solution.

Ores that are treated by dump leaching are leached with a sodium cyanide solution.

Ores that are treated in situ are--

1. Rubblized in-place.
2. Leached with a sodium cyanide solution.

Although it is possible to leach gold and silver with several solutions, all current operations use weak cyanide solution. Leach solutions are applied generally with sprays of either the oscillating or the fixed variety. After percolating down through the ore, the solution drains off and is collected in a holding pond. The gold and silver are then recovered from the pregnant leach solutions by precipitation with zinc dust or by adsorption onto activated carbon. Each method of metal recovery has its various advantages and disadvantages. Selection of one system over the other will depend on the specific conditions present at the leach operation.

LEACHING PROBLEM AND RESEARCH

Several problems hamper the broader application of leaching methods for recovering gold and silver. These problems are listed below:

1. Presence of clay or clay-sized particles, which retards leach solution percolation.
2. "Tight" or impermeable matrix, which reduces leach solution contact with the metal value in the matrix.
3. Inclement weather conditions that prohibit extended leaching activities.

4. Leach solution loss through evaporation.

5. Presence of refractory-type ores that inhibit cyanide leaching.

The Bureau of Mines has been active for many years in the development of new techniques for recovering and processing gold and silver ores. A major aim or effort has been to develop applied technology to help increase the domestic production of the vital minerals such as precious metals. Two current studies are--

1. Particle agglomeration techniques to improve percolation and recovery rates during heap leaching.

2. Feasibility study to evaluate in situ leach mining of gold and silver ores.

CONCLUSION

Gold and silver solution mining (leaching) operations have sprung up in many mining districts of the Western United States. Over 80 operations have been identified that have conducted tests or established commercial operations. The geology favoring leachable deposits seems to be in regions that have been subjected to folding, faulting, and volcanic activity. Leaching practices are offshoots from 70 or 80 years of conventional milling operations wherein gold and silver have been dissolved with cyanide. Instead of processing the ore through a complex mill circuit, leaching operators dissolve the metal directly from run-of-mine or crushed rock. Gold and silver are then recovered from the pregnant liquors using the traditional zinc precipitation process or by the relatively new charcoal adsorption process. Although gold and silver can be leached in situ or from waste rock dumps, heap leaching on specially prepared pads is the predominant method.

The main problems encountered in heap leaching operations are poor solution percolation due to high clay content in the ore and mineralogy that is detrimental to leaching reactions. Cold temperatures and lime buildup in the solution distribution system also can severely affect the economics of an operation. Bureau research on particle agglomeration offers intriguing possibilities for reducing poor percolation rates due to clay. The problems of possible refractory ores must be worked out in the laboratory in advance of the decision to leach and are not discussed herein. At least one company has experimented with submersible kerosene heaters for warming leaching solutions, and results to date have been promising. Conversion from lime to NaOH reduces lime buildup in distribution lines; several operators have experimented with Bagdad wigglers in additional attempts to minimize maintenance costs associated with lime buildup. These wigglers, named for their original use at the Bagdad copper mine, are easily constructed from 9-inch segments of thick-walled gum rubber tubing.

SELECTION OF LIXIVIANTS FOR IN SITU LEACH MINING

by

Daryl R. Tweeton[1] and Kent A. Peterson[2]

ABSTRACT

This paper provides information to assist in selecting a lixiviant (leach solution) for in situ uranium leaching. The cost, advantages, and disadvantages of lixiviants currently used and proposed are presented. Laboratory and field tests are described, and applications of geochemical models are discussed.

INTRODUCTION

Selection of the lixiviant is of critical importance to the success of an in situ leaching operation. The lixiviant affects not only the recovery of uranium and the cost of chemicals, but also the difficulty of meeting environmental regulations concerning restroation of ground water quality after leaching.

No data specifically on lixiviant selection have previously been made available to the public. Much research has been done by companies, but the results have usually been considered proprietary. However, useful literature is available on topics that are important parts of the lixiviant selection process. The chemistry of conventional milling is thoroughly discussed by Merritt.[3] The similarities in chemistry between milling and in situ leaching make this a very useful reference. Extensive column leaching studies were performed by Westinghouse Electric Corp. for the Bureau of Mines Salt Lake City Research Center.[4] The influence of various lixiviants on the difficulty of restoring the ground water quality after leaching is an important factor and is discussed in several publications.[5]

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[2] Geologist, Twin Cities Research Center, Bureau of Mines, Minneapolis, Minn.

[3] Merrit, R. C. The Extractive Metallurgy of Uranium. Colorado School of Mines Research Institute, Golden, Colo., 1971, 576 pp.

[4] Grant, D. C. In Situ Leaching Studies of Uranium Ores--Phase IV. BuMines Open File Rept. 52-79, 1978, 497 pp.; available from National Technical Information Service, Springfield, Va., PB 296 336/AS.

[5] Kasper, D. R., H. W. Martin, L. D. Munsey, R. B. Bhappu, and C. K. Chase. Environmental Assessment of In Situ Mining. BuMines Open File Rept. 101-80, 1979, 292 pp.; available from National Technical Information Service, Springfield, Va., PB 81-106783.

Thompson, W. E., W. V. Swarzenski, D. L. Warner, G. E. Rouse, O. F. Carrington, and R. Z. Pyrih. Groundwater Elements of In Situ Leach Mining of Uranium. Prepared for U.S. Nuclear Regulatory Commission, August 1978, 173 pp.; available from National Technical Information Service, Springfield, Va., NUREG/CR-0311.

Because of the importance of the subject to in situ uranium leaching, and because of the lack of previously published information, the Bureau of Mines has prepared Information Circular 8851 on this topic, which will be published later this year. This paper summarizes that Information Circular.

AVAILABLE LIXIVIANTS

Lixiviants that have been used for in situ uranium leaching include solutions of ammonium carbonate-bicarbonate, sodium carbonate-bicarbonate, carbon dioxide, and sulfuric acid. Potassium carbonate-bicarbonate is technically attractive but has been considered too expensive. Hydrochloric and nitric acids have been proposed for leaching carbonaceous ore. The carbonate-bicarbonate and sulfuric acid lixiviants contain an anion that will form a soluble complex with uranium in its +6 charge state. The cation does not directly affect the solubility of the uranium but is important because of its effect on permeability, cost, and ground water quality restoration.

An oxidizer is required to convert unoxidized uranium from its insoluble +4 charge state to its soluble +6 charge state. Oxidizers that have been used include oxygen, hydrogen peroxide, and sodium chlorate.

The costs, not including delivery, of chemicals used for making lixiviants are listed in table 1 in the units in which the market prices are commonly expressed and in dollars per kilogram. Costs were obtained from discussions with suppliers and leaching companies and from published prices in late 1980.[6] These units are not the most useful for comparing costs because they do not directly compare the cost of providing the significant component. For example, 1 kg of potassium carbonate provides less carbonate than 1 kg of ammonium carbonate.

TABLE 1. - Costs of chemicals used for making lixiviants

Chemical	Form	Cost, 100-pct basis	
		Per ton	Per kg
CO_2	Compressed and cooled liquefied gas[1]	\$60- \$200	\$0.07-\$0.22
NH_3	Ammonia fertilizer	200	.22
NaOH	Caustic soda, liquid, 50 pct at \$250/ton.	500	.55
Na_2CO_3	Soda ash	90	.10
$NaHCO_3$	Flakes or powder	240	.26
$NaHCO_3 \cdot Na_2CO_3 \cdot 2H_2O$	Sodium sesquicarbonate granules	96	.11
KOH	Caustic potash, liquid, 45 pct KOH at \$11/100 lb.	490	.54
K_2CO_3	Liquid, 47 pct K_2CO_3 at \$11,75/100 lb.	500	.55
$KHCO_3$	Granulated, technical-grade	280	.31
H_2SO_4	Liquid, concentrated, virgin	80	.09
HNO_3	Liquid, 58.5 to 68 pct HNO_3	120	.13
HCl	Liquid, 37 pct HCl at \$70/ton	190	.21
O_2	Liquid, at \$0.40 to \$0.60/100 cu ft of O_2 gas at 1 atm and 25° C (1 lb = 12.08 cu ft).	[2]97 [3]145	[2].11 [3].16
H_2O_2	Liquid, 50 pct H_2O_2 at \$0.29/lb.	1,160	1.28
$NaClO_3$	Powder or flakes	400	.44

[1] Depending on annual use. See text.
[2] Texas.
[3] Wyoming.

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[6] Chemical Marketing Reporter. Sept. 22, 1980, 43 pp.

To facilitate comparing the costs, tables 2, 3, and 4 present the cost per kilogram-mole and per pound-mole of alkaline lixiviants, acid lixiviants, and oxidizers, respectively. Table 2 lists the costs of bicarbonate and carbonate lixiviants separately to permit calculating the cost of a lixiviant containing any proportion of the two. These lixiviant costs were calculated using the costs of chemicals in table 1, so changes in those chemical prices will cause proportionate changes in the corresponding lixiviant costs. The cost of $80 per ton of carbon dioxide used in formulating table 2 is typical, but it can vary a great deal.

TABLE 2. -Summary of alkaline lixiviant costs, advantages, and disadvantages

Lixiviant	Cost			Advantages and disadvantages
	Per kg-mole	Per lb-mole	Ppm U_3O_8 paying[1] for 3 g/l anion	
Ammonium lixiviants:				
NH_4HCO_3	$7.63	$3.46	5.7	Little effect on permeability. Difficult to meet restoration requirements.
$(NH_4)_2CO_3$	11.38	5.16	8.6	
Sodium lixiviants:				
From soda ash:				
$NaHCO_3$	7.19	3.27	5.4	Relatively easy to meet restoration requirements. Can reduce permeability.
Na_2CO_3	10.52	4.77	8.0	
From caustic soda:				
$NaHCO_3$	25.93	11.76	19	
Na_2CO_3	47.98	21.76	36	
From sodium sesquicarbonate:				
$NaHCO_3$	8.67	3.93	6.5	
Na_2CO_3	17.60	7.98	13	
Potassium lixiviants:				
From caustic potash:				
$KHCO_3$	33.56	15.22	25	Little effect on permeability, should be relatively easy to meet restoration requirements. Expensive unless preceded by chloride preflush.
K_2CO_3	63.26	28.69	48	
From granules: $KHCO_3$	30.90	14.00	23	
From 47-pct K_2CO_3 solution:				
K_2CO_3	76.18	34.55	58	
CO_2	[2]3.88	[2]1.76	[2] [3]2.9	Cheap, little effect on permeability, easy to meet restoration requirements. Not effective in all deposits.

[1] Assuming $66/kg for U_3O_8 and neglecting recycling.

[2] Assuming $80/ton.

[3] For 3 g/l HCO_3.

TABLE 3. - Summary of acid lixiviant costs, advantages, and disadvantages

Lixiviant	Cost[1]			Advantages and disadvantages
	Per kg-mole (per kg-equiv wt)	Per lb-mole (per lb-equiv wt)	Ppm U_3O_8 paying[2] for 0.051 M solution[3] (0.102 $\overline{N}$ solution)[3]	
H_2SO_4....	\$8.65 (4.33)	\$3.92 (1.96)	6.7 (6.7)	Very effective in amenable deposits, restoration easier than with $(NH_4)_2CO_3$. Not usable in deposits with much $CaCO_3$, not selective for U.
HNO_3.....	8.33 (8.33)	3.78 (3.78)	6.4 (13)	Claimed to be effective for carbonaceous deposits. Not selective for U, dissolves Ra, requires cationic IX resin, difficult restoration.
HCl......	7.61 (7.61)	3.45 (3.45)	5.9 (12)	Claimed to be effective for clayey deposits. Not selective for U, dissolves Ra, requires cationic IX resin.

[1] Figures in parentheses expressed in term shown in parentheses in the corresponding boxheads.

[2] Assuming \$66/kg for U_3O_8 and neglecting recycling.

[3] Equivalent to 5 g/l H_2SO_4.

TABLE 4. - Summary of oxidizer costs, advantages, and disadvantages

Oxidizer	Cost			Advantages and disadvantages
	Per kg-mole	Per lb-mole	Ppm U_3O_8 paying[1] for 0.3 g/l O	
O_2:				
Texas..	\$3.40	\$1.55	0.48	Cheap. Must be injected downhole, can cause gas blockage near injection wells.
Wyoming	5.10	2.33	.72	
H_2O_2.....	43.48	19.72	12	Can be added to lixiviant above ground. Expensive.
$NaClO_3$...	46.97	21.30	4.4	Solubility does not depend on pressure. Na can reduce permeability. Cl can reduce ion-exchange resin efficiency.

[1] Assuming \$66/kg for U_3O_8 and neglecting recycling.

When assessing the significance of the chemical costs, it is useful to express them in terms of the parts per million U_3O_8 in solution that pays for the chemical costs of a typical strength lixiviant. Accordingly, table 2 includes the parts per million U_3O_8 required to pay for 3 g/l carbonate or bicarbonate, a typical concentration. Table 3 lists the parts per million U_3O_8 required to pay for acid molar and normal concentrations equivalent to 5 g/l sulfuric acid. The costs of equivalent normalities are included because they are comparisons of the costs of obtaining a selected pH. Table 4 lists the parts per million U_3O_8 required to pay for 0.3 g/l oxygen, which is a typical concentration, provided by each of the oxidizers. A value of $66 per kilogram ($30 per pound) is assumed for U_3O_8.

The parts per million U_3O_8 listed in tables 2-4 were calculated assuming no recycling of the lixiviants, and so are upper limits. Recycling was not included because it depends on site-specific factors. Discussions with leaching company personnel suggest that 60 to 90 percent of the lixiviant can be recycled at most sites. The parts-per-million values can be compared with the 17 to 200 ppm U_3O_8 in the pregnant solutions from successful operations. The comparisons can help avoid incorrect conclusions. For example, one might infer that sodium bicarbonate should not be used because it costs twice as much as an equivalent concentration of dissolved carbon dioxide. However, when recycling is considered, the cost difference is equivalent to only about 1 ppm U_3O_8 and so will have less impact than a very small difference in leaching efficiency.

Tables 2, 3, and 4 also summarizes the advantages and disadvantages of the various alkaline and acid lixiviants and the oxidizers, respectively.

METHODS OF TESTING LIXIVIANTS

The costs, advantages, and disadvantages previously presented provide only a general guide for lixiviant selection. To determine the suitability for a specific deposit, thorough laboratory and field testing is necessary.

Laboratory Tests

Both batch leach tests (sometimes called agitation leach tests) and column leach tests are used in selecting the lixiviant. Batch leach tests consist of placing the ore and lixiviant in a container, often a sealed flask, and gently agitating them. Although they do not simulate downhole conditions, they can show the relative rate and amount of uranium extraction with tested lixiviants and can give an indication of lixiviant and oxidant consumption. Obtaining meaningful results from oxidizer consumption tests requires special care to avoid oxidizing the ore before the test.

Column leaching tests simulate field conditions more closely than batch tests, but caution must still be used when extrapolating from laboratory to field. The contact between ore and lixiviant is more complete than in actual in situ leaching. Therefore, for both batch and column leaching, the measured consumption of lixiviant and oxidizer and the extraction of uranium should be viewed as upper limits of what might be expected in the field.

Column leaching tests can indicate permeability losses, but to obtain meaningful results, water from the formation should be used and the ore should be disaggregated and blended. Attempts to use intact cores in hopes of better simulating downhole conditions have not been satisfactory. Meaningful comparisons of lixiviants require similar cores, but cores vary considerably in permeability and uranium content.

Pilot Field Tests

A pilot-scale field test is conducted before starting commercial operation. It is needed not only as an aid to making the final choice of lixiviant, but also for evaluating well construction and completion techniques and for demonstrating restoration procedures.

Pilot-scale tests can be divided into two classifications. The first type is called push-pull, or huff-and-puff. The lixiviant is injected and recovered from the same well. There is some disagreement as to the value of push-pull tests. The second type can be called flow-through. The lixiviant is injected, flows through the formation, and is recovered from other wells, as it is in most commercial operations. Therefore, many consultants prefer the flow-through test.

Problems can occur that render a pilot field test useless as a guide in making the final choice of lixiviant. Problems that have occurred include the following:

1. Leaking casings.

2. Clogging of well screens or nearby formation.

3. Clogging of formation near a production well.

4. Reprecipitation of uranium.

GEOCHEMICAL MODELS

Geochemical models applied to in situ uranium leaching can assist in lixiviant selection. The models can be divided into two major categories. The first type of model, the equilibrium approach, is useful for describing numerous interactions of a complex system of aqueous species and solid phases. This type of model can be used to determine the reactions that are likely to occur within a given system, but it gives no information concerning the rates of the reactions.

The second type of model, the kinetic model, simulates the progress of kinetic reactions as a function of time and location. Because kinetic models cannot be used in selecting a lixiviant unless pertinent reaction rates are first determined through laboratory experiments, this report will concentrate on equilibrium modeling.

As of 1980, probably the most useful model for in situ uranium leaching is an updated version of the equilibrium program WATEQF.[7] The program requires as input a relatively complete chemical analysis of the solution of interest. A table of thermodynamic data for all reactions modeled by the program must also be read into the computer. WATEQF computes the state of saturation of the solution with respect to various minerals and amorphous solid compounds. The program compares the activity product of the ions involved in the appropriate reaction with the thermodynamic equilibrium constant for that reaction, and calculates the log of that ratio, which is termed the saturation index (S.I.). If S.I. is significantly less than zero, the solution is undersaturated with respect to that mineral. If S.I. is close to zero, then the reaction is close to equilibrium. If S.I. is greater than zero, the solution is supersaturated with respect to that mineral. This does not necessarily indicate precipitation, because solutions can remain supersaturated with respect to some minerals for a long time. Thus, the program is useful for predicting trends in solubility with changes in lixiviant composition, but cannot necessarily predict the concentrations that will be measured.

As of 1980, at least two companies are using WATEQF (as modified by Runnels) to assist in determining how the lixiviant composition should be changed to improve leaching. One company uses it to help select the most cost-effective lixiviant composition for dissolving the uranium minerals. The cost of a solution providing a given pH and Eh can be estimated, and the solubilities of the minerals can be predicted with WATEQF. Thus, for a given lixiviant cost, the program can help select the combination of pH and Eh maximizing solubility. Judgment is still required for balancing cost versus solubility, however.

WATEQF has also been used to predict whether solubilities will increase or decrease with changes in carbonate concentration, pH, or Eh. It is especially helpful in determining the probable cause and suggesting a cure when pilot tests are yielding much less uranium than expected. This company also uses WATEQF to predict the relative amounts of uranium species. Uranium as a monocarbonate complex will not load on anionic exchange resins, and so is undesirable. WATEQF predicts what fraction will be in monocarbonate, dicarbonate, and tricarbonate complexes. The program has also been used to predict fouling from minerals precipitating in pipes and to study restoration geochemistry.

SUMMARY

The selection of a lixiviant for in situ mining usually proceeds through three phases. First, general advantages and disadvantages of lixiviants are considered. These general considerations include technical, economic, and environmental factors. Currently, restoration of ground water quality is causing a movement away from ammonium carbonate-bicarbonate toward sodium

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[7] Runnels, D. D., R. Lindberg, S. L. Lueck, and G. Markos. Applications of Computer Modeling to the Genesis, Exploration, and In Situ Mining of Uranium and Vanadium Deposits. New Mexico Bureau of Mines and Mineral Resources, Socorro, N. Mex., Memoir 38, 1980, pp. 355-367.

bicarbonate and dissolved carbon dioxide. The cost of the oxidizer should be carefully considered, because it can exceed the cost of all the other chemicals.

Second, lixiviants that seem promising are tested with ore (cores) from the site to be leached. Laboratory batch and column leaching experiments measure leaching effficiency, consumption, and effect on permeability. These tests can be misleading if not conducted and interpreted with care.

Third, a pilot-scale field test is conducted. Proper well construction is vital to the success of this test. The test can be either the push-pull or the flow-through type. The former is cheaper, but the later simulates full-scale conditions more closely.

Computer modeling of the geochemistry can aid in the selection process. Such models are being used by at least two leaching companies to predict changes in solubilities associated with possible changes in lixiviant composition.

ADVANTAGES OF USING A CHLORIDE PREFLUSH BEFORE CARBONATE IN SITU LEACH MINING

by

Daryl R. Tweeton[1]

ABSTRACT

Laboratory experiments indicate that the consumption of potassium carbonate-bicarbonate can be greatly reduced if the ore is conditioned with potassium chloride before leaching. Because potassium chloride is relatively cheap, the cost of using potassium carbonate-bicarbonate is reduced to the extent that substituting it for ammonium carbonate-bicarbonate appears feasible. This substitution facilitates postleach restoration of ground water quality. Flushing the ore with a chloride solution before leaching also helps to reduce permeability losses from calcium carbonate precipitation.

INTRODUCTION

The restoration of ground water quality to the criteria set by regulatory agencies is difficult or impossible following leaching with ammonium carbonate-bicarbonate. Alternative lixiviants such as sodium carbonate-bicarbonate, dissolved carbon dioxide, and sulfuric acid have limitations resulting in their use not being feasible in many deposits. (These limitations are discussed in the paper titled "Selection of Lixiviants for In Situ Leach Mining.") Potassium carbonate-bicarbonate is environmentally and technically attractive, but has been considered too expensive to use. In late 1980, 1 kg-mole of potassium bicarbonate cost $31, whereas 1 kg-mole of ammonium bicarbonate cost $8.

Researchers at the University of Texas at Austin, funded through a Bureau of Mines contract, have developed a procedure that promises to greatly reduce the cost of using potassium carbonate-bicarbonate. The researchers primarily responsible for developing the procedure are Terry Guilinger, Michael Breland, and Robert Schechter.

THE CHLORIDE PREFLUSH

The procedure consists of flushing the ore with potassium chloride before leaching with potassium carbonate-bicarbonate. In most ore, much of the consumption of lixiviant is by cation exchange. Therefore, satisfying the cation exchange sites with potassium from the potassium chloride before leaching reduces the consumption of potassium carbonate-bicarbonate during leaching. Because potassium chloride is relatively cheap, $5 per kilogram-mole in late 1980, the cost of using potassium carbonate-bicarbonate is reduced.

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In laboratory experiments, the consumption of potassium carbonate-bicarbonate was reduced 83 percent. If the same reduction occurred when in situ leaching with potassium bicarbonate, and if it is assumed that any decrease in bicarbonate consumption requires an equal increase, on a molar basis, of the chloride preflush, then the effective cost of potassium bicarbonate is (0.17)($31) + (0.83)($5) = $9 per kilogram-mole. Thus, the effective cost of the potassium bicarbonate would be similar to the $8 per kilogram-mole cost of ammonium bicarbonate. This calculation suggests that substituting potassium carbonate-bicarbonate for ammonium carbonate-bicarbonate would be economically feasible. Of course, a thorough site-specific economic comparison should include not only material costs of the chemicals, but also factors such as freight, labor, and equipment for handling the chemicals, and the possible effects of chloride on the loading ability of resins.

The laboratory experiments indicated that an additional benefit was better maintenance of permeability. The permeability was often nearly twice as high during leaching following the chloride preflush as it was without the preflush. Maintaining permeability during laboratory tests was attributed to a reduction in calcium carbonate precipitation. Ammonium or potassium in high concentrations tends to drive calcium off clays by ion exchange. The calcium may be transported some distance in a supersaturated condition, but causes clogging when it precipitates. Calcium chloride is much more soluble than calcium carbonate, so calcium can be removed by the chloride preflush. This benefit could also be obtained if a sodium chloride preflush preceded a sodium carbonate-bicarbonate lixiviant in deposits where the sodium did not cause excessive clay swelling. The same experiments showed that the chloride preflush did not reduce the uranium recovery.

The calcium-rich solution produced during the chloride preflush may also be useful during restoration. Depending on the postrestoration limits set for potassium, it may be advisable to inject a solution of high ionic strength during part of the restoration flushing to facilitate removal of potassium by ion exchange. To avoid creating new restoration problems, the primary cation in the high-ionic-strength solution should be harmless and found in fairly high levels in natural ground water. Thus, the calcium-rich solution produced during the chloride preflush appears ideal for that purpose. A patent disclosure on the chloride preflush method has been filed in the Solicitor's Office, U.S. Department of the Interior, Washington, D.C.

RESTORING GROUND WATER QUALITY FOLLOWING IN SITU LEACHING

by

Daryl R. Tweeton[1]

ABSTRACT

To assist mining companies in planning for restoration of ground water quality following in situ uranium leaching, the Bureau of Mines funded the preparation of two reports. "Restoration of Groundwater Quality After In Situ Uranium Leaching" primarily describes options for disposing of the waste solution from restoration and provides engineering cost estimates. "Analysis of Groundwater Criteria and Recent Restoration Attempts After In Situ Uranium Leaching" summarizes restoration attempts, presents an empirical equation predicting the amount of ground water flushing required, and presents State and Federal permit requirements. This paper summarizes some of the information from those reports.

INTRODUCTION

When planning in situ uranium leaching, the restoration of groundwater quality is one of the areas of greatest uncertainty. To assist mining companies in such planning, the Bureau of Mines has funded the preparation of two reports.

The first report was completed in 1979 by Ford, Bacon, and Davis Utah, Inc., and is titled "Restoration of Groundwater Quality After In Situ Uranium Leaching." It primarily describes the various options for dealing with the large volumes of waste solution from restoration and presents engineering cost estimates. It also describes related geology, geochemistry, regulations, and several restoration attempts.

The second report was completed in 1981 by Resource Engineering and Development, Inc., and is titled "Analysis of Groundwater Criteria and Recent Restoration Attempts After In Situ Uranium Leaching." Volume I contains summaries of restoration attempts within the last 5 years, capital costs of disposal systems reported by operators, and an empirical equation that provides a guide as to the amount of ground water flushing required to meet restoration criteria. Volume II contains in situ leaching permit requirements, including restoration requirements, for Texas, Wyoming, New Mexico, Utah, Montana, Colorado, and South Dakota, and Federal requirements.

This paper summarizes some of the information in those reports. Those who want the complete contract reports should contact Daryl Tweeton at the Bureau of Mines in Minneapolis, Minn., 612-725-3468.

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DISPOSAL METHODS

The waste solution from in situ leaching and from postleach restoration can be disposed of in either a deep disposal well or an evaporation pond. Generally, deep disposal wells have been used in Texas and evaporation ponds in Wyoming.

Deep-Well Disposal

Injection of waste through a deep well into a zone that does not contain useful water offers the advantage that the waste is completely removed from the biosphere. Examples of disposal of waste solutions similar to that from an in situ leaching operation occur in a report on uranium mills in New Mexico[2] and in Union Carbide's permit for the Palangana Dome uranium plant.[3]

A deep-well disposal system includes equipment required to concentrate and condition the waste stream for injection and to transport the waste solution from the mining site to the injection well. Deep-well disposal is limited to waste solutions that will not plug the injection zone by the precipitation of solids in reactions between the solution and the matrix of the host aquifer. In some cases, precipitation can be prevented or reduced by adjusting pH or adding retardants such as sodium hexametaphosphate for calcium sulfate.

Summaries of the capital and operating costs are presented in tables 1 and 2. Capital costs are calculated for variations of each of the primary factors affecting a disposal well: injection rate, well depth, and drilling difficulty. The operating cost estimate is divided into the direct costs of power, chemicals, and operating and maintenance, and a concluding summary of operating costs that includes overhead expenses and fixed charges. Power costs are calculated for an average wellhead pressure of 260 psi. Chemical costs include acid for pH adjustment, polyphosphate to retard calcium sulfate deposition in the injection zone, and copper sulfate to control bacteria and fungi. Chemical additions are proportional to flow rate.

TABLE 1. - Deep-well disposal capital costs versus well depth and rock type, mid-1978 dollars

	Well capacity	
	200,000 gpd (single well)	1 million gpd (2 wells at 500,000 gpd each)
5,000-ft well depth:		
Average rock.......	1,202,000	3,485,000
Difficult rock.....	1,345,000	3,761,000
10,000-ft well depth:		
Average rock.......	1,538,000	4,148,000
Difficult rock.....	2,083,000	5,220,000
15,000-ft well depth:		
Average rock.......	2,001,000	5,069,000
Difficult rock.....	3,200,000	7,440,000

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[2] Lynn, R. D., and Z. E. Arlin. Anaconda Successfully Disposes Uranium Mill Waste Water by Deep Well Injection. Min. Eng., v. 14, July 1962, pp. 49-52.

[3] Union Carbide Corp. Permit for Subsurface Disposal of Industrial Waste, No. WDW-134. Texas Water Quality Board, Austin, Tex., Sept. 22, 1976.

TABLE 2. - Operating costs for deep-well disposal system

(5,000-ft well of average drilling difficulty)

	Capacity			
	200,000 gpd		1 million gpd	
	Cost per 1,000 gal[1]	Pct of total	Cost per 1,000 gal[1]	Pct of total
Direct costs:				
Power (injection pump, transfer pumps, ancillary loads)	$0.13	3	$0.13	5
Chemicals:				
pH adjustment	.33	8	.33	13
Sodium hexametaphosphate	.06	2	.06	2
Copper sulfate	.01	Neg	.01	Neg
Operating and maintenance:				
Operating labor	.09	2	.04	2
Operating supervision (15 pct of OL)	.01	Neg	.01	Neg
Maintenance and repairs (1 pct of TCI)	.20	5	.12	5
Laboratory charges (10 pct of OL)	.01	Neg	Neg	Neg
Total direct costs	.84	21	.70	28
Overhead costs:				
Plant overhead (60 pct of O&M)	.19	5	.10	4
Administrative (15 pct of O&M)	.05	1	.03	1
Total overhead	.24	6	.13	5
Total direct and overhead costs	1.08	27	.83	33
Fixed charges:				
Sinking fund payment (8 pct, 10-yr life)	1.38	35	.80	32
Interest (10 pct, 50-50 debt-equity)	1.00	25	.58	23
Insurance, taxes, miscellaneous (2.5 pct)	.50	13	.29	12
Total fixed charges	2.88	73	1.67	67
Total operating costs	3.96	100	2.50	100

Neg Negligible.
OL Operating labor.
O&M Operating and maintenance.
TCI Total capital investment.
[1] Mid-1978 dollars.

Solar Evaporation Ponds

The liquid waste from the leaching operation or from surface treatment facilities can be evaporated in a shallow pond with a large surface area. As evaporation occurs a sludge remains, which is an important disadvantage because there are stringent regulations governing the disposal of the sludge.

Summaries of capital and operating costs for solar evaporation are listed in tables 3 and 4. The cost for disposing of the sludge at the pond site by backfilling and sealing is included in the estimate. To estimate costs appropriate for in situ leaching, an initial grade of 1 percent and a pond lining of 10-mil PVC are assumed. Costs change for variation of feed capacity, net evaporation rate at the site, grade, and lining. (The contract report discusses available linings.) The fixed charges dominate, as would be expected for systems requiring extensive excavation and little operating labor. Expenses are roughly inversely proportional to th net evaporation rate.

TABLE 3. - Total capital investment for solar evaporation ponds, mid-1978 dollars

Net evaporation rate, in/yr	Pond system capacity	
	200,000 gpd	1 million gpd
40	3,010,000	15,148,000
30	4,018,000	20,221,000
20	6,037,000	30,380,000
10	12,108,000	60,929,000

TABLE 4. - Operating costs for solar evaporation pond system at 40-in/yr net evaporation rate

	Pond system capacity			
	200,000 gpd		1 million gpd	
	Cost per 1,000 gal[1]	Pct of total	Cost per 1,000 gal[1]	Pct o total
Direct costs:				
Power (pumps and ancillary loads)	$0.03	Neg	$0.03	Neg
Chemicals	0	0	0	0
Operating and maintenance:				
Operating labor	.03	Neg	.01	Neg
Operating supervision (15 pct of OL)	Neg	Neg	Neg	Neg
Maintenance and repairs (0.25 pct of TCI)	.10	2	.10	2
Laboratory charges (10 pct of OL)	Neg	Neg	Neg	Neg
Total direct costs	.16	3	.14	2
Overhead costs:				
Plant overhead (60 pct of O&M)	.10	1	.09	1
Administration (15 pct of O&M)	.03	Neg	.02	Neg
Total overhead costs	.13	2	.11	1
Total direct and overhead costs	.29	4	.25	4
Fixed charges:				
Sinking fund payment (8 pct, 10-yr life)	2.85	46	2.75	46
Interest (10 pct, 50-50 debt-equity)	2.06	33	1.99	33
Insurance, taxes, miscellaneous (2.5 pct)	1.03	17	.99	17
Total fixed charges	5.94	96	5.73	96
Total operating costs	6.23	100	5.98	100

Neg Negligible.
OL Operating labor.
O&M Operating and maintenance.
TCI Total capital investment.
[1] Mid-1978 dollars.

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SURFACE TREATMENT

The waste stream from leaching or from restoration can be sent directly to the disposal system (well or pond), or it can first be treated to produce two streams. One stream is purified water, and the other is a more concentrated brine carrying most of the dissolved solids. The advantages of the second method are that the purified water can be reused, thereby reducing the total consumption of water, and the disposal system does not need as large a capacity to receive the concentrated brine as to receive the total waste stream.

The surface treatment technique that has been used by in situ leaching companies is reverse osmosis. Other treatment methods that are potentially useful are described.

Reverse Osmosis

Reverse osmosis is a physical means of separating dissolved ions from an aqueous stream. An externally applied pressure in excess of the solution's inherent osmotic pressure forces water through a semipermeable membrane while the dissolved ions are rejected. A solution's inherent osmotic pressure is a function of the type of constituents, the ionic characteristics of the dissolved solids, and the relative and absolute concentrations of the solutes. A useful rule of thumb for in situ leaching solutions is that 1,000 mg/l dissolved ions requires approximately 10 psi of applied pressure.

Tables 5 and 6 summarize capital and operating costs, based on actual field systems and experience, as of mid-1978. The sizes of the field systems range from 10,000 to 1 million gpd. These reverse osmosis units incorporate a flexible mechanical design to maximize water recovery, pertinent instrumentation to monitor water quality and flow, a design to minimize membrane fouling and scaling, and a membrane cleaning system. These units are skid mounted and require only power and piping hookups. These prices do not include site engineering fees or freight costs. The operating costs include power, operation, maintenance, and chemicals. The cost assumptions are power at 2.5 cents per kilowatt-hour, membrane replacements required at a rate of 50 percent per 3 years, and a maintenance requirement from past experience. The estimate is based on labor and supervision for round-the-clock and round-the-week operation, with the reverse osmosis unit set up and producing at full capacity for 300 days per year.

TABLE 5. - Capital costs for reverse osmosis system, mid-1978 dollars

	Capacity (feed rate)	
	200,000 gpd	1 million gpd
Direct costs:		
Equipment unit[1] (membrane assembly, high-pressure pump, basic instrumentation	139,000	597,000
Peripheral equipment[1] (prefilters, surge tank, holding tank, water quality and flow instrumentation, pH control system, transfer pumps, piping, valves	97,000	358,000
Other direct costs (20 pct of equipment): Delivery costs, installation costs, site improvements, electrical hookups, miscellaneous	47,000	191,000
Total direct costs	283,000	1,147,000
Indirect costs (5 pct of direct costs): Engineering and supervision, construction expenses	14,000	57,000
Total direct and indirect costs	297,000	1,204,000
Contractor's fees (2 pct)	6,000	24,000
Total capital investment	303,000	1,228,000

[1] Basic cost data for equipment provided by L. J. Kosarek, Director of Systems Engineering Research and Development, El Paso Environmental Systems, El Paso, Tex. To convert basic data for product-water capacity to feedwater capacity, an operation with 85-pct water recovery is assumed.

TABLE 6. - Operating costs for reverse osmosis treatment

	Capacity (feed rate)			
	200,000 gpd		1 million gpd	
	Cost per 1,000 gal[1]	Pct of total	Cost per 1,000 gal[1]	Pct of total
Direct costs:				
Power:				
Feed pump power....................	$0.13	11	$0.13	13
Ancillary (10 pct at feed pump):				
Transfer pumps, booster pumps, chemical feeders, instrumentation, and lighting................	.01	1	.01	1
Chemicals..........................	.06	5	.06	6
Operating and maintenance:				
Operating labor....................	.02	2	Neg	Neg
Operating supervision (25 pct of OL)................................	Neg	Neg	Neg	Neg
Maintenance material and labor[2] (includes membrane replacement)..	.11	9	.11	11
Total direct costs............	.33	28	.31	31
Overhead costs:				
Plant overhead costs (60 pct of O&M)	.11	9	.08	8
Administrative costs (15 pct of O&M)	.03	2	.02	2
Total overhead costs..........	.14	11	.10	10
Total direct and overhead costs........................	.47	39	.41	41
Fixed charges:				
Sinking fund payment (8 pct, 10-yr life).............................	.35	29	.28	28
Interest (10 pct, 50-50 debt-equity)	.25	21	.20	20
Insurance, taxes, miscellaneous (2.5 pct).........................	.13	11	.10	10
Total fixed charges...........	.73	61	.58	58
Total operating costs.........	1.20	100	.99	100

Neg Negligible.

OL Operating labor.

O&M Operating and maintenance.

[1] Mid-1978 dollars.

[2] Evaluated from information provided by L. J. Kosarek, Director of Systems, Engineering Research and Development, El Paso Environmental Systems, El Paso, Tex.

Other Treatment Methods

Other methods that are described in the contract report include electrodialysis, distillation, ion exchange, foam separation, and freeze separation.

Electrodialysis can be viewed as a combination of reverse osmosis and ion exchange. Ions pass through semipermeable membranes under the influence of an electric field. In a typical design, membranes, spacers, and electrodes are stacked and held together by end plates much like a plate and frame filter. Spacing is usually about 0.1 inch, and spacers are arranged to provide a tortuous flow path. Stacks range from 0.5 to 2,400 square meters of membrane area. A large stock can desalt 150 gpm at 20- to 50-percent salt removal. Practical systems use two to six stages. Electrodialysis is more expensive than reverse osmosis. A cost estimate from a supplier of electrodialysis equipment indicated a total operating cost of $2 to $3 per 1,000 gallons.

Distillation appears to be prohibitively expensive, four to five times the cost of reverse osmosis. The high cost is partly due to the high energy requirements. Similarly, ion-exchange treatment costs two to five times as much as reverse osmosis.

Water purification by freezing has not been applied to in situ leaching, but the process is claimed to have the potential for low costs, high water recovery, and effective contaminant rejection. The basis of the process is the principle that when ice is frozen from an aqueous solution of salts, the ice is a distinct and purer phase of water. The ice excludes most of the salts from its crystal structure. Costs for freeze separation have been estimated to be 20 to 40 percent greater than costs for reverse osmosis treatment for small flow rates, and potentially 20 to 40 percent less than costs for reverse osmosis for high flow rates.

SUMMARIES OF RESTORATION ATTEMPTS

The results of restoration attempts conducted at five operations in Texas and one in Wyoming (Irigaray) are summarized in table 7, prepared in the summer of 1980. With the exception of the commercial restoration at Intercontinental Energy Corp.'s Pawnee property, these restoration attempts may all be described as relatively small field tests. Several of these companies are, however, preparing for large-scale restoration of their mined-out areas.

TABLE 7. - Summary of restoration attempts

Company	IEC		Mobil		Mobil		WMC		WMC		WMC		WMC	
Site	Pawnee		O'Hern		O'Hern		Irigaray		Irigaray		Irigaray		Bruni	
Leaching reagents	$NH_3HCO_3 + H_2O_2$		NH_3HCO_3 + oxidant		O_2 + natural HCO_3^-		NH_3HCO_3 + oxidant		NH_3HCO_3 + oxidant		NH_3HCO_3 + oxidant		$NH_3HCO_3 + H_2O_2$	
Type of attempt	Mined out area		Field test		Field test		Field test		Field test		Field test		Field test	
Area involved	75 by 250 ft		20 by 20 ft		20 by 20 ft		25 by 25 ft		25 by 25 ft		25 by 25 ft		25 by 25 ft	
Patterns involved	5		1 with 2 wells		1 with 2 wells		1		1		1		1 with 2 holes	
Restoration process used[1]	GWS + RO + spraying		GWS + cation elution		GWS		Clean RO H_2O recycle		Chemical restoration + cation elution.		Cation elution + RO GWS.		Clean H_2O recycle (RO).	
Site-specific factors	Shallow deposit, thin deposit, leach chemicals, low clay content.		High clay content		High clay content		High clay content		High clay content		High clay content		High montmorillonite clay content.	
Pore volumes used	[1]2		6.2		3.67		0.5		NA		15.2		NA	
	Restoration levels achieved versus target restoration levels													
	Achieved	Target	Achieved	Target	Achieved	Target	Achieved	Target	Achieved	Target	Achieved	Target	Achieved	Target
NH_3-N, mg/l	167	0.01	88	1.9	NA	NA	105	<1.0	<35	<1.0	33	<1.0	90	1
U_3O_8, mg/l	2.4	2.0	NA	NA	1.8	2.45	5.7	0.098	<2	0.098	1	0.098	9	<0.5
TDS, mg/l	911	903	BB	NA	941	844	NA	NA	NA	NA	NA	NA	NA	NA
Mo^{++}, mg/l	2.8	1.0	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA
Cl^-, mg/l	107	250	211	205	202	203	120	10.75	NA	NA	113	10.75	494	623
Ca^{++}, mg/l	80	200	BB	NA	18	8.6	NA	NA	NA	NA	28	([2])	27	105
Conductivity, µmho/cm	1,899	1,310	1,730	1,450	1,586	1,470	1,600	([2])	NA	NA	NA	NA	758	([2])

Company	WMC		WMC		U.S. Steel		U.C.C.		WMC		WMC	
Site	Bruni		Irigaray		Clay West		Palangana		Bruni		Irigaray	
Leaching reagents	$NH_3HCO_3 + H_2O_2$		NH_3HCO_3 + oxidant		NH_3HCO_3 + oxidant		$NH_3HCO_3 + H_2O_2$		$NH_3HCO_3 + H_2O_2$		$NAHCO_3 + O_2$	
Type of attempt	Field test		Field test		Field test		Field test		Field test		Field test	
Area involved	25 by 25 ft		25 by 25 ft		0.92 acre[3]		50 by 50 ft		25 by 25 ft		0.8 acre	
Patterns involved	1 with 2 holes		1		13		1		1 with 2 holes		11	
Restoration process used[1]	Cation elution + strip NH_3		GWS		GWS		GWS		GWS		GWS	
Site-specific factors	High montmorillonite clay content.		High-CA clay content		Low and variable permeabilities, normal fault, ground water migration.		Poor permeabilities--clay zones.		High montmorillonite clay content.		High clay content	
Pore volumes used	NA		NA		[3]8.4		6.5		NA		[1]5	
	Restoration levels achieved versus target restoration levels											
	Achieved	Target	Achieved	Target	Achieved	Target	Achieved	Target	Achieved	Target	Achieved	Target
NH_3-N, mg/l	17	1	123	<1.0	12-71	0.5	16	1.4	300	1	BB	NA
U_3O_8, mg/l	0.5	0.5	12.3	0.098	2-21	0.5	NA	NA	NA	NA	12	0.098
TDS, mg/l	NA	NA	712	793	NA	NA	NA	NA	NA	NA	700	NA
Mo^{++}, mg/l	NA	NA	<0.002	0.0028	3-22	1.0	NA	NA	NA	NA	NA	NA
Cl^-, mg/l	500	623	229.9	10.75	261-770	120-400	NA	NA	600	558-687	BB	NA
Ca^{++}, mg/l	127	105	<0.002	<0.005	NA	NA	NA	NA	80	74-135	BB	NA
Conductivity, µmho/cm	NA	NA	1,950	([2])	NA	NA	1,040	1,200	4,500	2,275-2,693	NA	NA

BB Below baseline.
GWS Ground water sweeping.
NA Not available.
RO Reverse osmosis.
TDS Total dissolved solids.

[1] Restoration ongoing at present time.

[2] Test is ongoing. Final value not available.

[3] Original leach area. Ground water migration caused contaminated area to spread to 3.6 acres.

Several different processes have been used in these restoration attempts. At the Pawnee site, Intercontinental Energy Corp. treated recycled ammonia leach solution aboveground by spraying and reverse osmosis. Mobil Oil Corp. tested several methods at the O'Hern site for flushing the ammonia from clays, including ground water sweeping and cation elution, and also tried a non-ammonia leach process. U.S. Steel Corp. has tested ground water sweeping at an old in situ leach pilot plant area at the Clay West property. U.S. Steel's method of disposing of several pore volumes in a deep disposal well and then discharging a treated stream to surface waters appears to have considerable merit. Ground water sweeping was also tested by Union Carbide Corp. in a small test at the Palangana site. Extensive ground water sweeping and cation elution has been done by Wyoming Mineral Corp. at both the Irigaray and the Bruni operations. Wyoming Mineral Corp. was testing ground water sweeping of an ore zone leached with sodium carbonate-bicarbonate and oxygen.

The flushing requirements in table 7 indicate how much ground water displacement is needed to achieve a given degree of restoration at that site. This gives operators an idea of the magnitude of the restoration problem and provides a basis for sizing solution disposal and treatment facilities and for establishing restoration schedules.

The restoration testing indicates that it is extremely difficult, if not economically and technically impossible under existing operating conditions and with present restoration technology, to reduce ammonia and aquifer solutions to the levels set by State regulatory agencies. Complete restoration, as defined by these agencies, may require 50 to 100 pore volumes or more if an ammonia leach process has been used. Each of the three major companies involved in in situ uranium leaching (Mobil, U. S. Steel, and Wyoming Mineral Corp.) has changed or is changing its major operations from ammonia to non-ammonia leach solutions.

The nonammonia testing that has been done by Mobil and by Wyoming Mineral Corp. indicates that without the adsorption of ammonia by clays, restoration is faster and more complete than when ammonia is used in leaching. However, it may still be relatively difficult to restore parameters such as uranium, molybdenum, total dissolved salts, and conductivity to the levels set by State regulatory agencies.

Ground water restoration appears to be a bigger problem than was thought earlier. Field testing has shown that "complete restoration," as defined by the State regulatory agencies, has not been attained with reasonable degrees of flushing at any of these sites.

COSTS REPORTED BY OPERATORS

The intent was to obtain the costs of actual restorations and then compare these costs with estimates in the earlier study. However, the available cost information was primarily capital costs of disposal wells and evaporation ponds. Operating costs were not available because the operators had performed little restoration of mined-out areas. They felt that it was too early to accurately estimate operating costs.

The capital costs of several deep disposal wells drilled in Texas during the past few years are shown in table 8. Possible reasons for the large variation in costs follow: Companies having low estimates may not have the same ancilliary pretreatment facilities included in their estimates, corrosion-proof equipment may be used in the case of the higher estimates, and some companies may not include the cost of idle pretreatment equipment that they intend to use. Comparing these costs with the estimates in the earlier study shows that the estimates are consistent with those for the Union Carbide and Wyoming Mineral Corp. wells, and are higher than the others.

TABLE 8. - Disposal well costs reported by in situ leaching operators in Texas

Company	Well depth, ft	Maximum flows per well, gpm	Ancillary equipment cost	Total well cost[1]
Intercontinental Energy Corp..................	4,000	50	NA	$300,000- 350,000
Mobil Oil Corp..........	4,500-5,000	100-150	$150,000	650,000
Union Carbide Corp.....	5,700	100	NA	1,200,000
U.S. Steel Corp........	4,500	200-250	200,000	500,000
Wyoming Mineral Corp. (Lamprecht Site)......	6,000	200	NA	[2]1,100,000

[1] Includes ancillary pretreatment equipment, pumps, ponds, etc.

[2] Does not include cost of ponds.

The capital costs of Wyoming Mineral Corp.'s evaporation ponds are listed in table 9. The estimates in the earlier study indicated that a 200,000-gpd pond capacity with a 35-in/yr evaporation rate costs $2,878,000, or $37,250 per acre. The actual field costs per acre are thus higher in this instance than the estimates.

TABLE 9. - Capital costs for WMC's evaporation ponds in Texas and Wyoming

Site	Pond size, acres	Pond evaporation rate, gpm	Evaporation rate, in/yr	Cost per acre
Bruni....................	3.5	6.3	35	$65,000
Lamprecht...............	8.9	16	35	65,000
Irigaray.................	12	36	58	80,000

CASE HISTORY OF A PILOT-SCALE ACIDIC URANIUM IN SITU LEACHING EXPERIMENT

by

Michael T. Nigbor,[1] William H. Engelmann,[2] and Daryl R. Tweeton[3]

ABSTRACT

The Bureau of Mines, in cooperation with the Rocky Mountain Energy Co., constructed wells, analyzed water samples, and otherwise assisted in a pilot-scale in situ uranium leaching experiment at the company's Nine-Mile Lake site near Casper, Wyo. The experiment is unique in that it is believed to be the first pilot-scale operation to complete the leaching-restoration cycle using sulfuric acid instead of the more common carbonate-bicarbonate leachant. This report summarizes activities at that site, including geochemical data from startup to restoration and comparisons between laboratory and field experiments.

Sulfuric acid proved to be an effective leachant. Restoration was successful but required extended flushing. The pH was the last parameter to return to baseline, requiring about 350 days. This was longer than predicted in laboratory simulations. This report is a brief summary of a more complete Report of Investigations, entitled "Case History of a Pilot-Scale Acidic in Situ Uranium Leaching Experiment," which is expected to be available in early 1982.

INTRODUCTION

The in situ mining operation, known as the Nine-Mile Lake Site, was located about 16 km (10 miles) north of Casper, Wyo. Activity centered on a roll-front uranium deposit at about the 165-meter (500-foot) depth located in the Teapot Sandstone Formation. Permeability is high (0.98 darcy), and the content of acid consumers is low (less than 0.1 percent); therefore, the site was considered ideal for experimentation with sulfuric acid in situ leaching.

It has been reported that the common ammonium carbonate-bicarbonate lixiviants are very difficult to flush from the ore body after leaching.[4] This is because ammonium ions become attached to clays in the formation at ion

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[1] Mining engineer.
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[3] Research physicist.
All authors are with the Twin Cities Research Center, Bureau of Mines, Minneapolis, Minn.

[4] Thompson, W. E., W. V. Swarzenski, D. L. Warner, G. E. Rouse, O. F. Carrington, and R. Z. Pyrih. Groundwater Elements of In Situ Leach Mining of Uranium. Final report prepared for U.S. Nuclear Regulatory Commission, NUREG/CR-0311, August 1978, 173 pp.

exchange sites and resist removal attempts. Another common lixiviant, sodium carbonate-bicarbonate, can reduce formation permeability by swelling clays. Sulfuric acid was selected for use at this site because the rate of dissolution is higher,[5] because it avoids the problems associated with the above two lixiviants, and because the formation is low in acid consumers.

Laboratory batch leaching tests conducted on ore from Nine-Mile Lake indicated that sulfuric acid was a much more cost effective lixiviant than

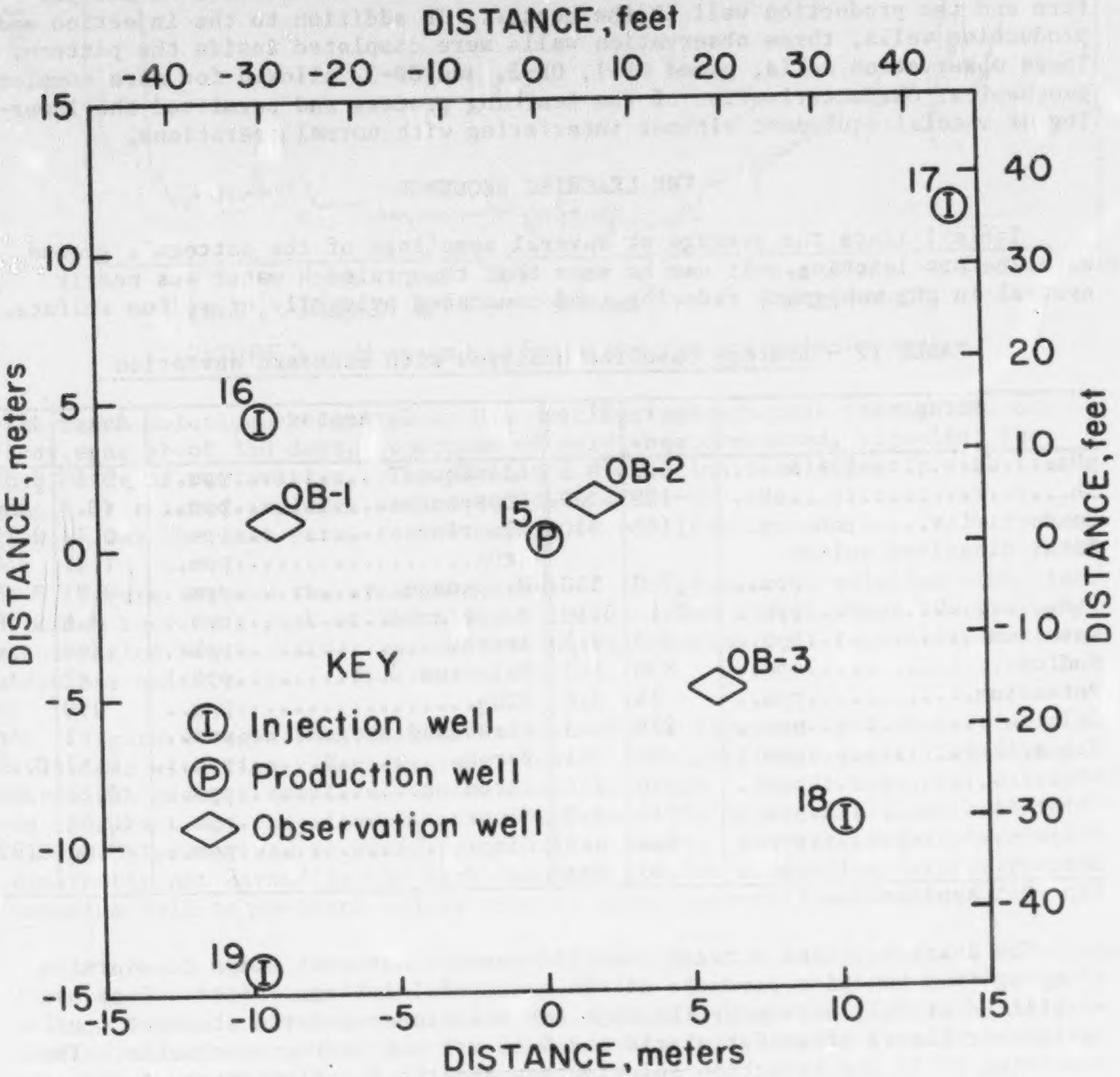
DISTANCE, feet
-40
-30
-20
-10
0
10
20
30
40
DISTANCE, meters
15
10
5
0
-5
-10
-15
DISTANCE, feet
40
30
20
10
0
-10
-20
-30
-40
16
17
18
19
15
OB-1
OB-2
OB-3
I
P
KEY
Injection well
Production well
Observation well
-15
-10
-5
0
5
10
15
DISTANCE, meters

FIGURE 1. - Plan view of pattern 2 at Nine-Mile Lake.

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[5] Merritt, R. C. The Extractive Metallurgy of Uranium. Colorado School of Mines Research Institute, 1971, 576 pp. (pp. 68 and 90).

the two carbonate-bicarbonate lixiviants. Later field results indicated that these laboratory tests gave misleadingly favorable results and that sulfuric acid offered little cost savings, if any. Potential environmental benefits, rather than cost savings, appear to be the major advantages of sulfuric acid.

THE TEST LEACH PATTERN

Figure 1 shows a plan view of pattern 2, the subject of this report. It is a five-spot pattern with four injection wells at the corners of the pattern and the production well in the center. In addition to the injection and production wells, three observation wells were completed inside the pattern. These observation wells, named OB-1, OB-2, and OB-3, allowed for more complete geochemical characterization of the leaching process and permitted the lowering of special equipment without interfering with normal operations.

THE LEACHING SEQUENCE

Table 1 lists the average of several samplings of the pattern's ground water before leaching. It can be seen that the preleach water was nearly neutral in pH, was quite reducing, and consisted primarily of sodium sulfate.

TABLE 1. - Average baseline analyses with standard deviation

Parameter	Average	±1 σ	Parameter	Average	±1 σ
pH..............-log(H+)..	6.7	0.3	Aluminum..............ppm..	0.13	0.05
Eh.....................mV..	-120	200	Phosphorus...........ppm..	<0.2	NAp
Conductivity.....µmho/cm..	4,100	510	Fluorine.............ppm..	0.77	0.25
			Iron.................ppm..	1.07	0.4
Total dissolved solids ppm..	4,300	550	Manganese............ppm..	0.31	0.18
V_3O_8..................ppm..	0.23	0.10	Molybdenum...........ppb..	0.8	0.18
Vanadium.............ppm..	0.5	0.2	Arsenic..............ppb..	<40	NAp
Sodium...............ppm..	830	145	Selenium.............ppb..	<2	NAp
Potassium............ppm..	14	3.6	HCO_3..................ppm..	290	30
Calcium..............ppm..	270	43	Dissolved oxygen.....ppm..	1	NAp
Magnesium............ppm..	92	31	Boron................ppm..	0.67	0.40
SO_4..................ppm..	2,510	244	Chromium.............ppm..	<0.01	NAp
Chlorine.............ppm..	46	4.3	Copper...............ppm..	<0.01	NAp
Silicon..............ppm..	4.2	4.0	Zinc.................ppm..	0.02	0.02
Mercury..............ppm..	<.01	NAp			

NAp Not applicable.

The leach solution strength was increased in several steps to minimize clogging from reaction products at the start of leaching. After values had stabilized at full strength, the injected solution consisted of about 4 g/l (grams per liter) of sulfuric acid and 0.10 vol-pct hydrogen peroxide. The resulting pH in the injection solution was about 1.8. Flow rates at the production well varied but averaged about 113 l/m (30 gpm).

Figure 2 shows the injection and production solution pH's during the life of the test. The pH dropped gradually to steady values at 50 days. The pH of

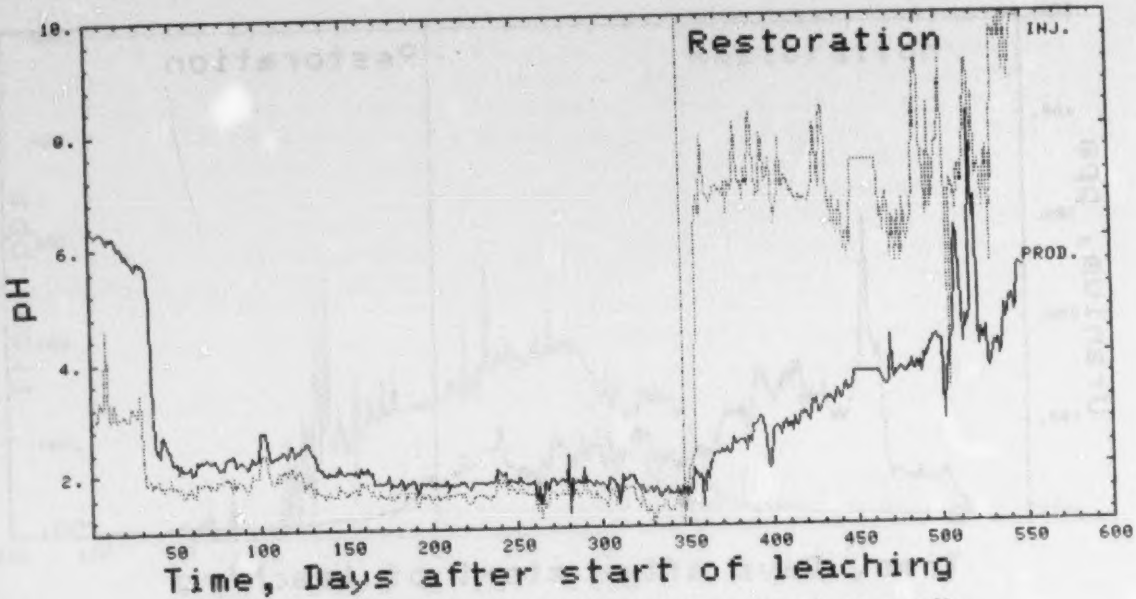

FIGURE 2. - pH versus time for the injection and production well.

the leach solution changed about 0.4 unit during each pass through the ore body. At about 350 days, injection of acid was terminated, signaling the beginning of restoration. The pH in the production rose linearly with time until it reached preleach values 300 days after the start of restoration. The pH was the last parameter to return to baseline.

Figure 3 shows the uranium content of the production solution with time. Uranium rose to a peak of 300 ppm and then leveled off at about 100 ppm until restoration began. After restoration began, uranium fell to preleach concentrations quickly.

Figure 4 shows the vanadium content of the injection and production solutions with time. Vanadium was present in significant quantities in the formation and was mobilized by the leach solution. Vanadium was not extracted in the plant and was allowed to recirculate. The only control was a 19-l/m (5-gpm) bleed stream to waste (evaporation pond). The extraction plant was apparently not harmed by the high vanadium content of the leach solution, and vanadium fell to preleach values quickly after restoration commenced.

Figure 5 shows the iron content of the injection and production solutions. Iron plays an important role in the oxidation of uranium to the soluble +6 state. Figure 6 shows a record of the conductivity of the injection and production solutions. Note that the conductivity fell quickly to near baseline conditions after the start of restoration.

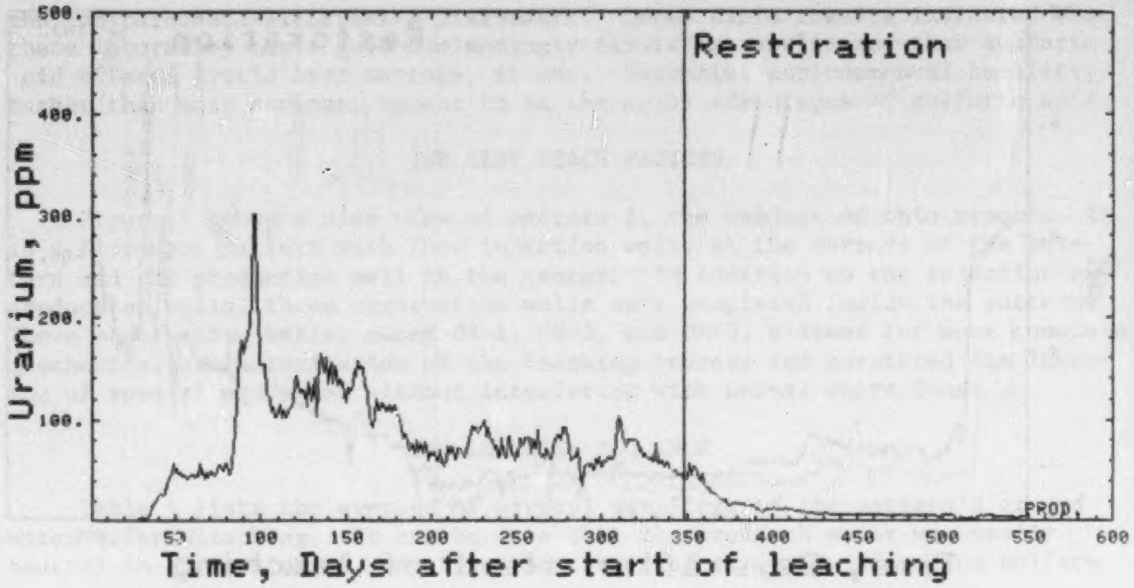

FIGURE 3. - Uranium versus time for the injection and production wells.

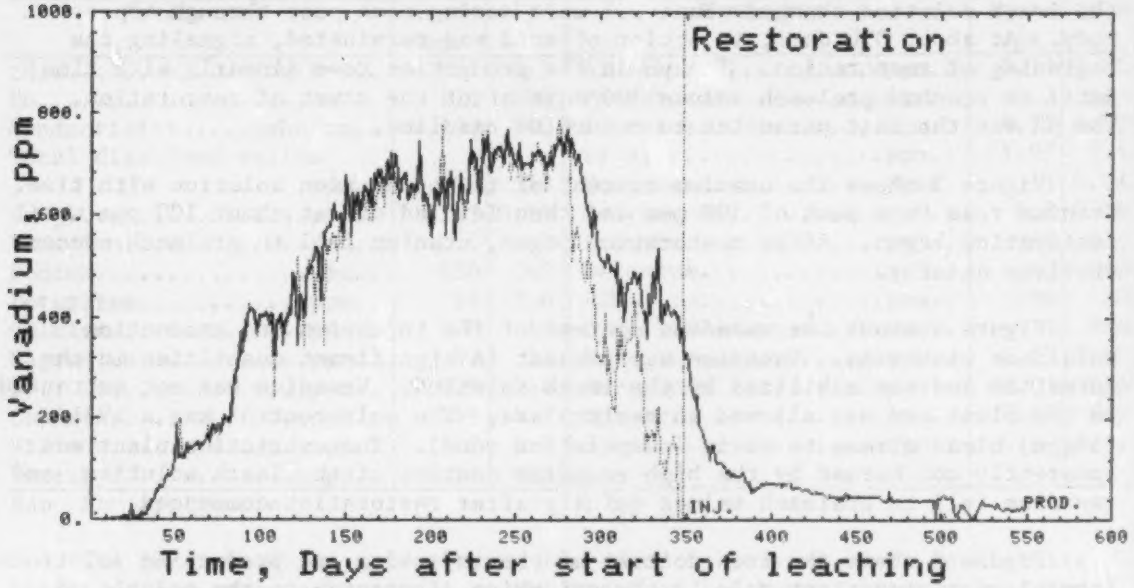

FIGURE 4. - Vanadium versus time for the injection and production wells.

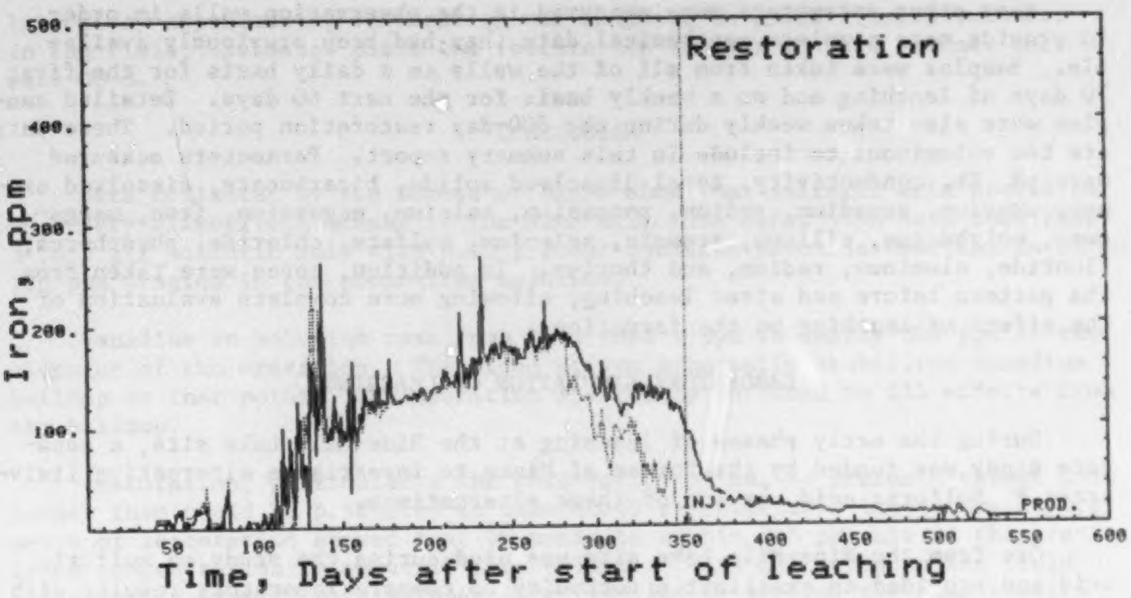

FIGURE 5. - Iron versus time for the injection and production wells.

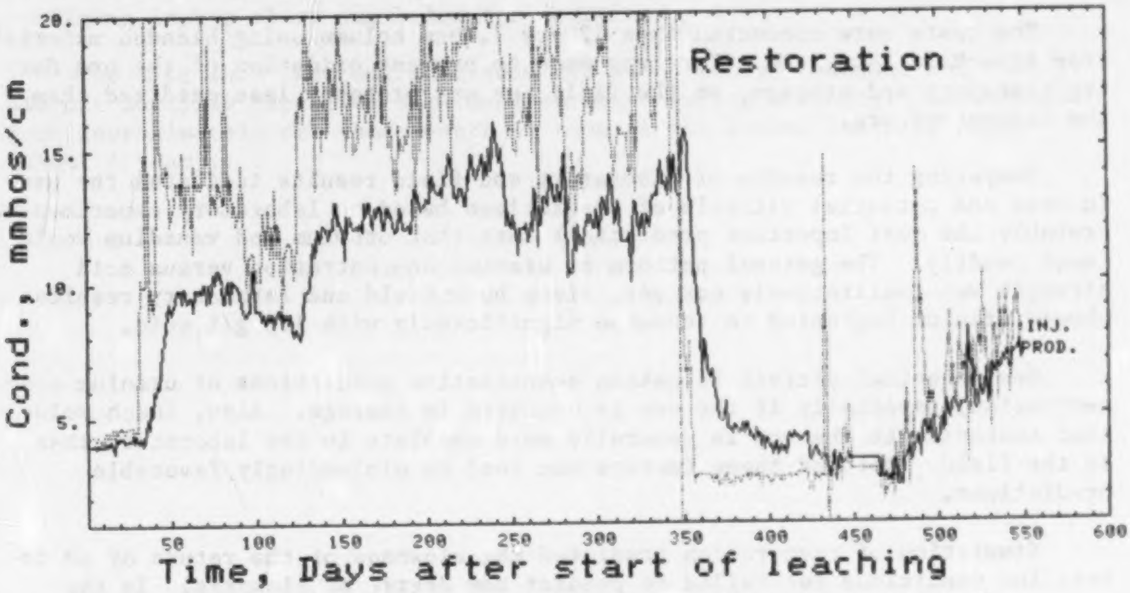

FIGURE 6. - Conductivity versus time for the injection and production wells.

Many other parameters were measured in the observation wells in order to provide more complete geochemical data than had been previously available. Samples were taken from all of the wells on a daily basis for the first 70 days of leaching and on a weekly basis for the next 60 days. Detailed samples were also taken weekly during the 300-day restoration period. These data are too voluminous to include in this summary report. Parameters measured were pH, Eh, conductivity, total dissolved solids, bicarbonate, dissolved oxygen, uranium, vanadium, sodium, potassium, calcium, magnesium, iron, manganese, molybdenum, silicon, arsenic, selenium, sulfate, chloride, phosphorus, fluoride, aluminum, radium, and thorium. In addition, cores were taken from the pattern before and after leaching, allowing more complete evaluation of the effect of leaching on the formation.

LABORATORY SIMULATION OF LEACHING

During the early phases of leaching at the Nine-Mile Lake site, a separate study was funded by the Bureau of Mines to investigate alternative lixiviants.[6] Sulfuric acid was one of these alternatives.

Ore from the Nine-Mile Lake site was used during the study on sulfuric acid and provided an excellent opportunity to compare laboratory results with field results. Such comparisons are vital for determining how to employ laboratory experiments for predicting field results.

The tests were conducted in a 120-by 7.6-cm column using blended material from Nine-Mile Lake. No effort was made to prevent oxidation of the ore during transport and storage, so the field ore was probably less oxidized than the laboratory ore.

Comparing the results of laboratory and field results indicates the usefulness and potential pitfalls of predictions based on laboratory experiments. Probably the most important predictions were that uranium and vanadium would leach readily. The general pattern of uranium concentration versus acid strength was qualitatively correct, since both field and laboratory results showed uranium beginning to increase significantly with 1.5 g/l acid.

One potential pitfall is making quantitative predictions of uranium concentration, especially if the ore is oxidized in storage. Also, leach solution contact with the ore is generally more complete in the laboratory than in the field. Both of these factors can lead to misleadingly favorable predictions.

Simulation of restoration predicted the slowness of the return of pH to baseline conditions but failed to predict the degree of slowness. In the laboratory experiments, restoration was complete in about 13 pore volumes. In the field, estimates place the restoration at over 20 pore volumes near well OB-3.

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[6] Sundar, P. S. In Situ Leaching Studies of Uranium Ores. Phases I through III. BuMines Open File Rept. 140-77, 1977, 392 pp.; available for consultation at the Bureau of Mines libraries in Minneapolis, Minn., Denver, Colo., and Salt Lake City, Utah; at the Central Library, U.S. Department of the Interior, Washington, D.C.; and from the National Technical Information Service, Springfield, Va., PB 272 717/AS.

CONCLUSION

Data collected by the Bureau of Mines show that sulfuric acid proved to be a very effective leachant at the Nine-Mile Lake site. The data show that 3 to 5 g/l sulfuric acid with 0.10 percent hydrogen peroxide resulted in 80 to 120 ppm uranium in the production solution.

Vanadium in solution rose from less than 1 ppm to nearly 800 ppm at the midpoint of the operation. The bleed stream apparently stabilized vanadium buildup at that point. The operation apparently suffered no ill effects from the buildup.

Restoration, particularly the restoration of pH, to preleach values took longer than could be predicted in laboratory experiments. Laboratory experiments of restoration showed that pH would be within 0.5 pH unit of the preleach concentration in about 13 pore volumes. Data collected in the field suggest that over 20 pore volumes were required to achieve the same results near well OB-3. This difference may be due to a number of factors, including permeability variations in the ore and leach solution contact with the shale confining layers above and below the deposit.

The data show that sulfuric acid was an effective leachant and that it did not mobilize excessive hazardous elements during leaching. Restoration took approximately the same length of time as the active leaching phase (about 1 year).

LABORATORY AND FIELD TESTING OF DRILLING FLUIDS TO DETERMINE HOW THEY AFFECT SANDSTONE PERMEABILITY

by

Jon K. Ahlness,[1] Donald I. Johnson,[2] and Daryl R. Tweeton[3]

ABSTRACT

The Bureau of Mines conducted laboratory and field experiments to determine the amount of permeability reduction in mineralized sandstones after exposure to different drilling fluids. Both polymer and bentonite drilling fluids were laboratory tested. The bentonite fluids resulted in the most permeability reduction in sandstone cores cut from samples collected at an open pit uranium mine. The fluid that resulted in the least permeability reduction was an hydroxyethyl cellulose polymer fluid. The greatest permeability reduction of the polymers came from guar-gum-based fluids and a synthetic polymer. Five polymer fluids were tested with simulated drill cuttings added to represent field conditions. The least permeability reduction in these tests was obtained from a multipolymer-blend fluid. A field experiment was then undertaken to compare two polymer fluids for drilling in situ uranium leaching wells. For this test, the polymer fluid with the best (multipolymer blend) laboratory results was compared with a commonly used polymer fluid (guar gum) that gave poorer laboratory results. When injection rates for the four wells drilled with the guar gum were compared with those for the four drilled with the multipolymer blend, no statistically significant difference was seen.

INTRODUCTION

A common problem for in situ leaching operations is low well injection rates. This is caused by low permeability in the formation near the well, which can result from the drilling process and is influenced by the type of drilling fluid used. The effects of different drilling fluids on sandstone permeability were the subject of a study done by the Bureau of Mines. Laboratory tests with several drilling fluids were conducted on sandstone cores. Based on the results of these tests, two drilling fluids were selected for field testing. The field test consisted of drilling a total of eight injection wells at an in situ uranium site and comparing the injection rates of the four wells drilled with each fluid.

Formation damage occurs by two methods. The first is the blocking of the pore openings in the wellbore due to a buildup of fine particles on the

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[3] Research physicist.

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hole wall. The second method of damage involves drilling fluid filtrate. This fluid moves from the well into the formation carrying fine particles until they lodge and block pore openings. The filtrate can also affect water-sensitive clays in the formation, causing them to swell. The result of these occurrences is the narrowing or plugging of pore spaces through which fluids may flow, thereby decreasing the permeability.

The drilling fluids applicable to drilling in situ leaching wells fall into two categories, bentonites and polymers. Bentonite is an inorganic gel-forming clay colloid, with the predomonant clay mineral being montmorillonite. This material is readily dispersible in water and forms a permanent viscous suspension which is thixotropic. It controls filtrate loss to the formation by forming an impermeable wallcake of clay particles on the wellbore. A polymer is a molecule formed by the union of two or more identical smaller molecules, the resulting compound having a molecular weight larger than that of, and chemical properties different from those of, any of the original components. Most polymers are derived from organic materials, although some synthetic polymers are available. Some common types of organically based polymers are guar gum, xanthum gum, carboxymethyl cellulose (CMC), and hydroxyethyl cellulose (HEC), along with various combinations and blends. Polymers control filtration loss by forming a network of polymer chains on the wellbore.

Previous related research on drilling fluids showed that guar gum drilling fluids reduced permeability by a factor of as much as four when injected into high-permeability sandstone. Polymer-based drilling fluids such as polyurethane or hydroxyethyl cellulose, with calcium carbonate bridging material, were found to be least damaging.

LABORATORY TESTS

Sandstone Core Samples

Sandstone samples were collected from an open pit uranium mine near Bill, Wyo. They were taken from newly exposed waste material from the pit floor. The quartz sandstone was relatively "clean," with the clay size fraction being less than 2 percent. The accessory minerals in the clay size fraction were identified as chlorite, muscovite, and sericite. Cores were cut approximately 2.54 cm (1 inch) in diameter and 2.54 cm (1 inch) long using air as the drilling medium. The length was limited because the sandstone was quite friable, especially in the coarser grain sizes. There was a large variation of grain size between cores that resulted in a wide range of initial permeabilities.

Test Apparatus

The laboratory drilling fluid test equipment consisted of a permeability test cell, two drilling fluid tanks, a brine tank, a breaker tank, and the tubing, valves, and fittings necessary to transport and the control fluids from the tanks to the cell. All components of the test apparatus were made of stainless steel. Nitrogen pressure was used to circulate the fluids to the cell. The permeability test cell (fig. 1) accommodated 2.54-cm (1 inch) diameter cores up to 10.2 cm (4 inches) in length.

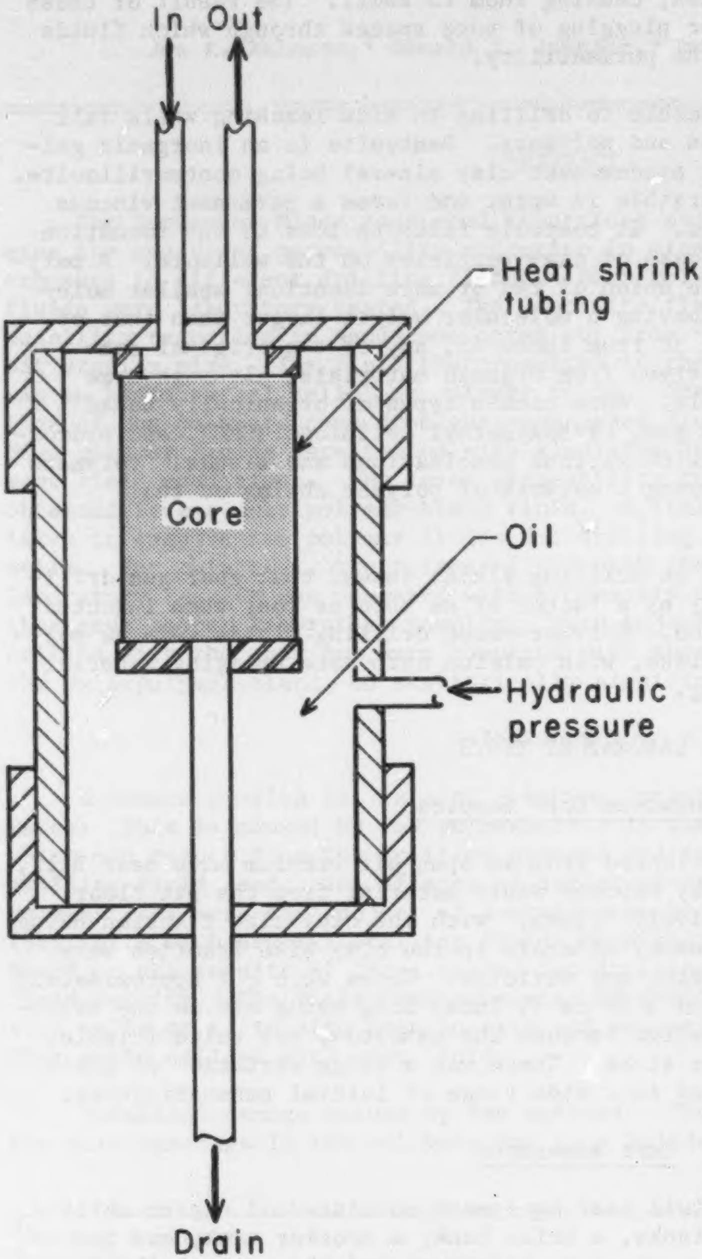

FIGURE 1. - Permeability test cell.

EXPERIMENTAL PROCEDURE

Drilling fluids were laboratory-tested in both "clean" and "dirty" (solids added) conditions. The dirty fluids were tested in an attempt to simulate the condition in which they are used in the field. The fluids were mixed in 12-liter (3.2 gallon) batches with a small electric mixer. Mixing was done for a minimum of 1 hour to allow the fluids to fully viscosify (hydrate). Dirty fluids were made by mixing Rev-Dust (a low-grade bentonite material) into a fully hydrated clean fluid. Mixing was continued for 30 minutes after the addition of the Rev-Dust. The dirty fluid was then allowed to stand overnight to let the excess solids settle out in the mixing container. The settled solids were dried and weighed to determine the remaining solids content of the dirty fluid. The fluids were then transferred to the drilling fluid tanks of the test apparatus.

Each fluid was used for a series of tests over a period of 5 to 12 days. Formaldehyde was added as a preservative to the polymer fluids that were susceptible to natural breakdown of viscosity due to bacterial actions. Even with this precaution, however, breakdown with time did occur in some fluids, resulting in decreasing viscosity from one test to the next.

Each test was run according to the following format:

1. Core mounted in permeability test cell.
2. Initial permeability test.
3. Circulation of drilling fluid.
4. Circulation of breaker (if any).
5. Overnight breakdown time.
6. Backflush with brine.
7. Final permeability test.

A core sample was evacuated in a vacuum chamber in a beaker of 3-percent NaCl (sodium chloride) brine. The brine was used to inhibit the hydration of any swelling clays which may have been present. The core was then mounted in shrink tubing, and the cell was pressurized to 2068 kPa (300 psi) for confinement. An initial permeability test was run by forcing brine through the core at 345 kPa (50 psi) and measuring the length of time required to collect 50 ml (0.013 gallon).

The drilling fluid to be tested was then circulated past one face of the core at 345 kPa (50 psi) for 1 hour to simulate dynamic downhole conditions that are present during drilling. After the drilling fluid was circulated, the appropriate breaker (if any) mixed in brine was circulated past the core face at 345 kPa (50 psi). An overnight breakdown time was then allowed. When a breaker was not recommended for a fluid, brine was circulated instead of a breaker. For some of the breakerless tests, an overnight rest was used for consistency with the breaker tests; in others, the rest of the test followed immediately.

The next step of the test procedure was to force brine through the core in the reverse direction (backflush). This backflushing was done at a pressure 517 kPa (75 psi) for 10 minutes to simulate well development by pumping. At this point a second (final) permeability test was run, which concluded the test.

LABORATORY TEST RESULTS

Clean Drilling Fluids

Seven different polymer fluids, a bentonite, and a bentonite-polymer combination were tested in their clean state. The amount of each fluid used and whether or not a breaker was recommended were determined from manufacturer's literature. Guar gum fluids from three different manufacturers were tested in clean and dirty conditions.

The test data are summarized in table 1. Test results are given in the form of average return permeability. This is the ratio of the final permeability to the initial permeability, given as a percent, which is the percentage of the original permeability remaining after exposure to the drilling fluid.

TABLE 1. - Summary of clean fluid tests

Drilling fluid	Number of tests	Fann viscosity, cp at 300 rpm	Average return permeability, pct	Standard deviation, pct
Guar gum 1...............	14	44	17	6
Multipolymer blend.......	9	44	36	9
Xanthum gum..............	12	17	44	24
Hydroxyethyl cellulose (HEC)..................	7	60	47	10
Organic ploysaccharide...	8	37	38	24
Bentonite-HEC............	6	43	6	6
Bentonite-HEC[1]..........	7	43	9	6
Bentonite[1]...............	9	6	27	14
Guar gum 2...............	5	29	23	5
Synthetic polymer........	5	16	5	3

[1] 1-day test.

The highest average return permeabilities were achieved from the HEC (47 percent) and the xanthum gum (44 percent) fluids. The xanthum gum results, however, were the most variable, with a standard deviation of 24 percent. The lowest average return permeabilities were obtained from the synthetic polymer (5 percent), the bentonite-polymer combination (6 and 9 percent), the two guar gum fluids (17 and 23 percent), and straight bentonite (27 percent). Two groups of tests were run on the bentonite-polymer fluid. Six tests were run with the overnight wait, and averaged 6-percent return permeability, and seven were run in one day which resulted in a slightly higher average return permeability of 9 percent.

Dirty Drilling Fluids

Six different polymer fluids were tested with simulated drill cuttings (Rev-Dust) added. The same amount (637 grams; 1.4 pounds) was added to each 12-liter (3.2 gallon) batch of fluid. This amount should have resulted in each fluid having a 5-percent-solids content. However, settlement occured when the fluid was allowed to stand overnight, resulting in some variability.

The test procedures, fluid mixing, and use of breakers were the same as for the clean fluid tests. The test data are summarized in table 2. The highest average return permeability was obtained from the multipolymer blend (43 percent). The lowest average return permeability results were from guar gum 3 (6 percent), xanthum gum (7 percent), and the synthetic polymer (7 percent).

TABLE 2. - Dirty drilling fluid summary

Drilling fluid	Percent solids	Fann viscosity, cp at 300 rpm	Number of tests	Average return permeability, pct	Standard deviation, pct
Guar gum 1............	4.36	39	5	26	14
Multipolymer blend....	4.91	48	5	43	6
Hydroxyethyl cellulose (HEC)................	3.96	60	5	25	16
Guar gum 3............	2.27	39	5	6	5
Xanthum gum...........	5.00	21	5	7	2
Synthetic polymer.....	2.92	13	4	7	4

FIELD TEST

Two polymer drilling fluids were compared in the field by drilling eight injection wells in a uranium sandstone formation at an in situ leaching site and then measuring the injection rate of the wells. The two fluids selected were guar gum 1 and the multipolymer blend. The guar gum was chosen because it is commonly used, and the multipolymer blend because it gave the best results in the laboratory tests. Each fluid was used to drill four wells. The well pattern and the fluid used to drill each well are shown in figure 2.

All the wells were constructed in the same manner, the only difference being in the drilling fluids. Each well was started with a 0.14 meter (5-1/2-inch) pilot hole drilled to the top of the sandstone formation. This hole was then reamed to 0.19 meter (7-3/8-inches) cased and cemented. The ore zone was then underreamed to a diameter of 0.28 meters (11 inches), a screen was set, and airlifting was done for well development. The viscosity of the underreaming fluids was measured with a Marsh Funnel and ranged from 32 to 40 seconds.

An injection test was then run simultaneously on all eight wells to determine if there was a difference in injection rates between the wells drilled with guar gum 1 and those drilled with the multipolymer blend. The test was done by injecting ground water at a constant rate of approximately 19 liters per minute (5 gpm) into each well and monitoring the resulting pressure heads (water level) in each well. Approximately 150 liters per minute (40 gpm) was pumped from the production well during the test. Pressure transducers were used for monitoring the head levels in the wells.

The injection test was started 5 days after completion of the last well and was run for 78 hours. The head level increase at the end of the test and other wellfield data are shown in table 3. The data show that all the inner wells have higher head levels (lower injection rates) than the corresponding outer well. This occurrence was independent of which drilling fluid was used in drilling the well. This indicates that the well pattern or the formation geology greatly affected the test results. When the injection rates of the four guar gum wells are compared with those of the four multipolymer blend wells, no statistically significant difference can be found.

TABLE 3. - Wellfield data and final head increase

Hole No.[1]	Underreaming fluid	Marsh funnel viscosity, seconds	Final head increase	
			Meters	Feet
58.........	Multipolymer blend.	33	24	78
59.........	Guar gum...........	32	48	157
60.........	Multipolymer blend.	35	10	32
61.........	Guar gum...........	33	19	63
63.........	Multipolymer blend.	33	6	19
64.........	Guar gum...........	34	5	15
65.........	Multipolymer blend.	34	20	67
66.........	Guar gum...........	40	8	25

[1] See figure 2.

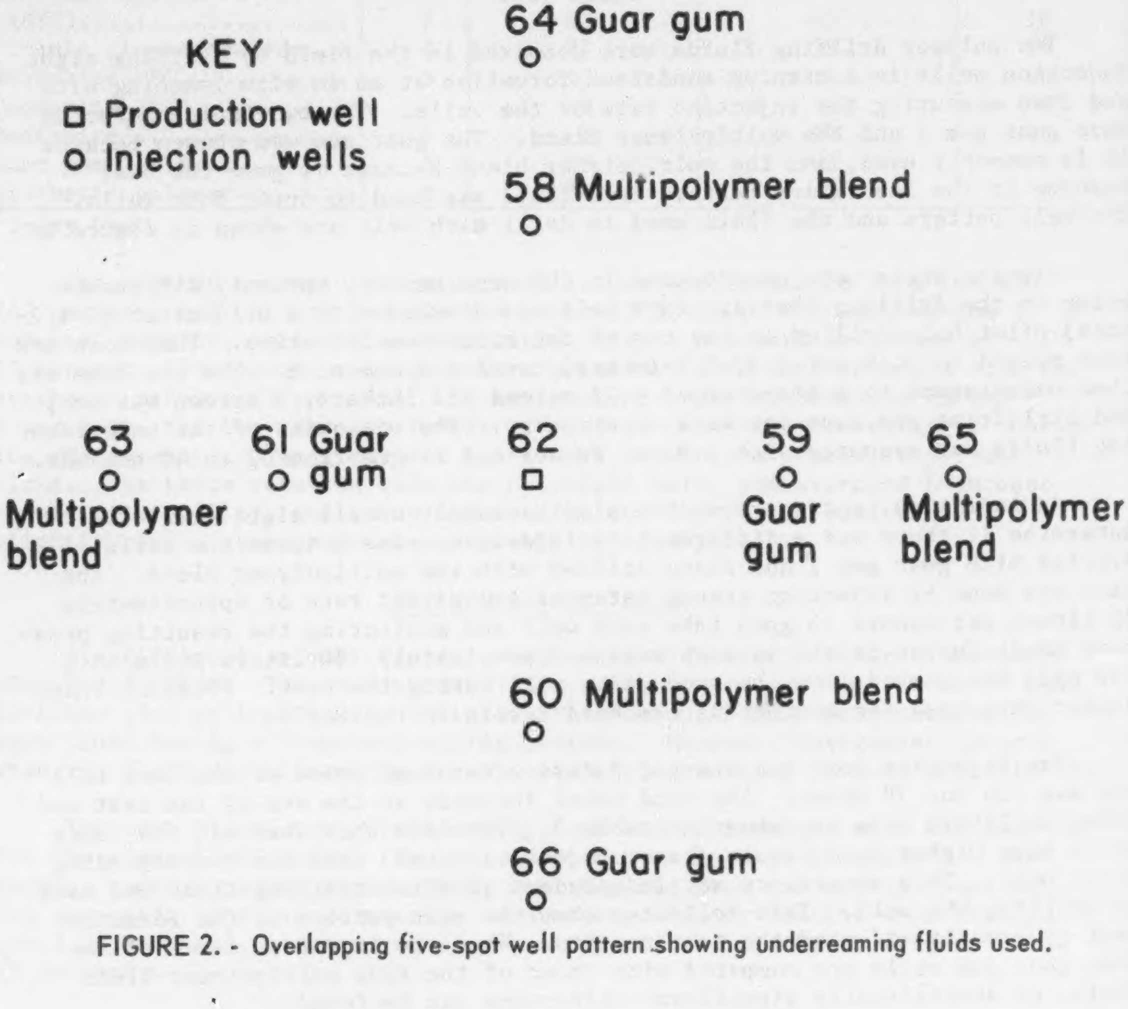
64 Guar gum
KEY
□ Production well
○ Injection wells
58 Multipolymer blend
63 Multipolymer blend
61 Guar gum
62
59 Guar gum
65 Multipolymer blend
60 Multipolymer blend
66 Guar gum

FIGURE 2. - Overlapping five-spot well pattern showing underreaming fluids used.

CONCLUSIONS

It was found from the laboratory tests that there were significant differences in the permeability damage caused by different types of drilling fluids. The HEC and multipolymer blend polymer fluids resulted in the highest average return permeabilities; bentonite, guar gum, and synthetic polymers resulted in the lowest. When guar gum and multipolymer blend drilling fluids were compared under identical field drilling conditions, however, no significant difference could be determined from injection rates for in situ uranium leaching wells.

APPLICATIONS OF GEOPHYSICAL RESISTANCE MEASUREMENTS TO IN SITU LEACHING

by

Daryl R. Tweeton[1]

ABSTRACT

Geophysical resistance and resistivity systems were tested to determine their applicability for indicating the movement of leach solution. Measuring the resistance between wells appeared promising, as well-to-well resistance dropped significantly when leach solution replaced ground water. Galvanic resistivity measurements using surface electrodes were less reliable, as the results were strongly influenced by factors other than the movement of leach solution. Audio magnetotelluric measurements were very susceptible to interference from powerlines.

INTRODUCTION

A means of inferring the pattern of underground movement of leach solution during in situ leaching would be helpful in at least two different situations. The first is in determining if the leach solution moves in the desired uniform pattern during injection. The second is in detecting the start of an excursion.

Accordingly, the Bureau of Mines funded a research contract, awarded to Westinghouse Electric Corp., to test the ability of several geophysical measurement systems to indicate the change in resistance or resistivity as leach solution replaces ground water. Details of the tests are in the contractor's final report to the Bureau. This paper summarizes that report. Those wanting a copy of that report should contact Daryl Tweeton at the Bureau's Twin Cities Research Center in Minneapolis (612 725-3468). Results were also published as

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[1] Research physicist, Twin Cities Research Center, Bureau of Mines, Minneapolis, Minn.

an AIME preprint[2] and the full report[3] is available from The National Technical Information Service for $14.

FIELD TESTS

Field tests were conducted in 1979 at an in situ uranium leaching operation in Wyoming. The ore zone was about 80 meters deep and 3 meters thick. Figure 1 shows the arrangement of test locations within the site. The most promising technique, measuring the resistance between wells, was tested in wells GI 20, 21, 22, 41, 42, and 43. The intent was to measure the resistance between the center well and each of the six corner wells before leaching began, and then daily after the start of injection of leach solution at the corners.

The probe configuration is shown in figure 2. A known current flows from I_1 to I_2, and the resulting voltage difference between V_1 and V_2 is measured. The configuration is similar electrically to four-terminal arrays used for galvanic resistivity with surface electrodes. However, the downhole probes have the important advantages that most of the current passes through the ore zone. The casing is nonconductive, so current flows only through the screened section. Thus the measured resistance depends primarily on factors within the ore zone. With surface electrodes, the ore zone is only a small fraction of the volume of earth affecting the measurements.

Figure 3 summarizes the results. The data show the resistance across the pattern in two directions instead of showing the resistance between the center well and each corner well because the current electrode in the center well showed a high resistance to current flow part of the time. Apparently, the drawdown was much greater than expected, and the current probe was out of the water during pumping. Therefore, resistance between injection wells across the pattern were measured.

Measurements were made on October 20 and 23, before leaching began. From October 23 to November 4, the field was being soaked with oxygenated ground water. That lowered the resistance somewhat. Injection of leach solution

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[2] Kehrman, R. F., A. J. Farstad, and D. R. Tweeton. Use of Resistivity Measurements To Monitor Lixiviant Migration During In Situ Uranium Leaching. Pres. at Fall Meeting, Soc. Min. Eng., AIME, Minneapolis, Minn., Oct. 22-24, 1980, SME Preprint 80-338, 10 pp.

[3] Kehrman, R. F. Detection of Lixiviant Excursions With Geophysical Resistance Measurements During In Situ Uranium Leaching. (Final Report, Contract J0188080 with Westinghouse Electric Corp., December 1979, BuMines Open File Rept. 5-81, 1981, 156 pp.; available for consultation at the Bureau of Mines libraries in Albany, Oreg., Avondale, Md., Boulder City, Nev., Denver, Colo., Pittsburgh, Pa., Reno, Nev., Rolla, Mo., Salt Lake City, Utah, Spokane, Wash., Tuscaloosa, Ala., and Twin Cities, Minn., at the DOE facilities at Carbondale, Ill., and Morgantown, W. Va.; at the National Mine Health and Safety Academy, Beckley, W. Va.; at the Office of Surface Mining Library, South Interior Building, Washington, D.C.; at the Central Library, U.S. Department of the Interior, Washington, D.C.; and from the National Information Service, Springfield, Va., PB 81-171324.

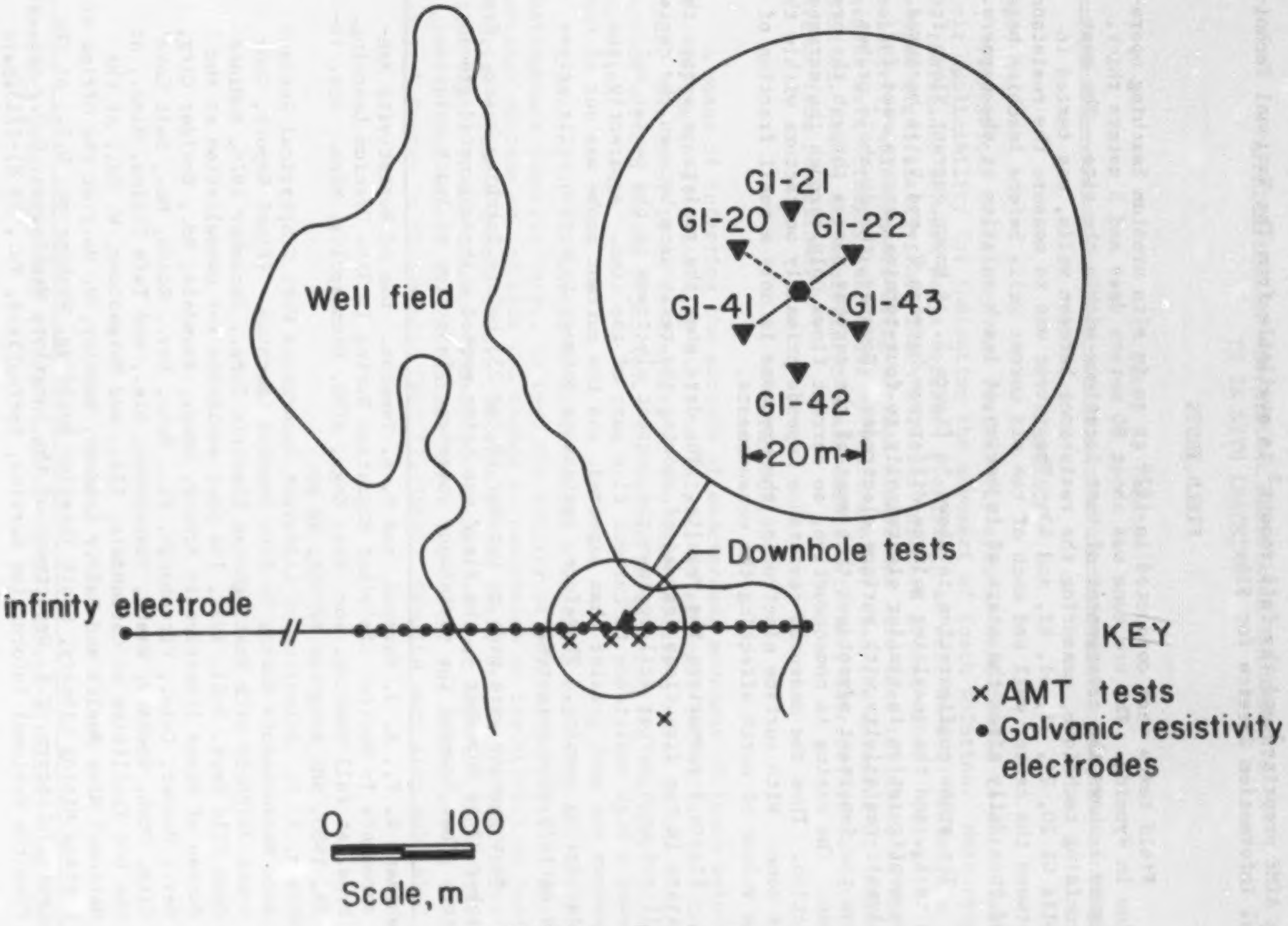
Well field
GI-21
GI-20
GI-22
GI-41
GI-43
GI-42
20 m
Downhole tests
infinity electrode
KEY
× AMT tests
• Galvanic resistivity electrodes
0
100
Scale, m

FIGURE 1. - Test locations.

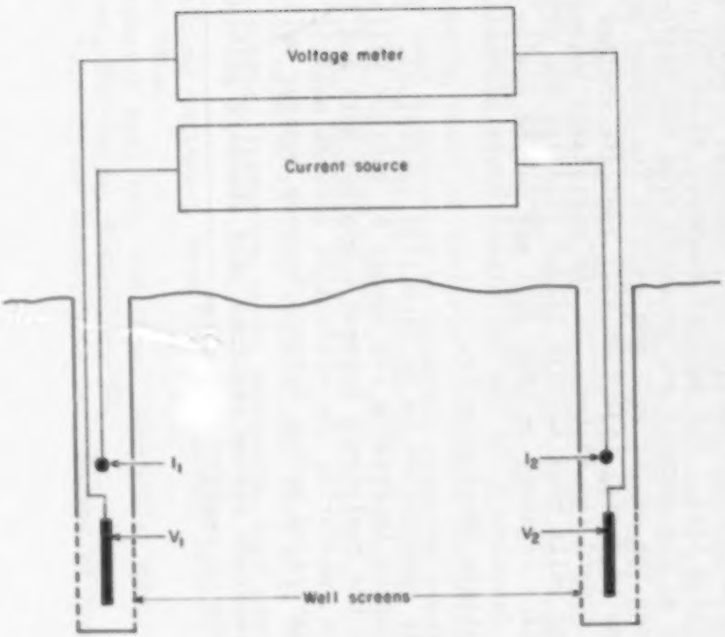
Voltage meter
Current source
I_1
V_1
I_2
V_2
Well screens

FIGURE 2. - Method of implementing borehole resistance monitoring techniques.

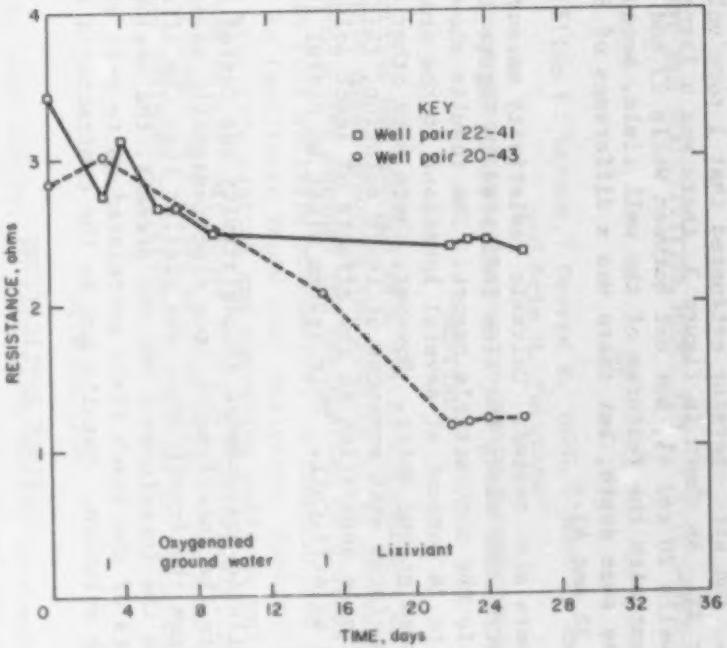

FIGURE 3. - In-hole monitoring results.

began on November 5. The site operator estimated that 1 pore volume had been injected by November 11. As shown in figure 3, there was a large decrease in resistance between wells 20 and 43, but not between wells 22 and 41. These results are consistent with the features of the well field, because wells 20 and 43 were at the same depth, but there was a difference of 6 meters depth between wells 22 and 41.

Other systems were also tested. Galvanic resistivity measurements with surface electrodes were made along the line indicated in figure 1. Details of the tests are given in the contractor's report. The results showed decreases in resistivity of 9 to 14 percent at several locations in the ore zones where leach solution replaced ground water. However, there were other changes in resistivity, not associated with movement of leach solution, that were almost as large. Thus, reliable separation of the effects of leach solution from the other effects whould be difficult. This system might be useful in very shallow deposits.

Audio magnetotelluric measurement of resistivity was tested and found to be very susceptible to the interference from electromagnetic waves from powerlines. The effect from the interference was greater than the effect from the leach solution. When the interference was not present, the measured resistivities in various parts of the leach field correlated quite well with the known distribution of leach solution. Details are in the contractor's report.

POSSIBLE APPLICATIONS

As an example of a possible application, consider a seven-spot pattern, as was used in the field test. For several reasons, leach solution would probably be injected at the corners and produced from the center. Measuring the resistance between the center well and each of the corner wells before injection and then daily during injection will indicate whether leach solution is moving uniformly toward the center. If the resistance between the center and one of the corner wells remains high longer than for the other corners, then the leach solution injected in that well may not be moving toward the center as desired. Conversely, if the resistance drops unusually quickly, there may be a hydrologic short circuit.

Another possible application is in detecting excursions. To detect excursions sooner without increasing the number of monitor wells, one could periodically measure the resistance between a monitor well and the nearest injection well. Also, measuring the resistance between monitor wells may help to detect a narrow excursion moving outward between monitor wells.

The downhole electrode system requires only standard galvanic resistivity instrumentation, which is readily available.

COST AND SENSITIVITIES MODEL FOR IN SITU LEACH MINING

by

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and Orin M. Peterson[4]

ABSTRACT

In situ mining has emerged as a viable alternative to conventional mining techniques. This paper presents the results of an assessment of uranium in situ leach mining costs through the application of process engineering and discounted cash flow analysis procedures. A computerized costing technique was developed to facilitate rapid cost analyses. Application of this model will generate mine life capital and operating costs as well as solve for economic production in cost per pound of U_3O_8. Conversely, rate of return may be determined subject to a known selling price. The data bases of the cost model have been designed to reflect variations in Texas and Wyoming site applications.

INTRODUCTION

During the past decade, in situ leach mining of uranium has emerged as a viable third alternative to conventional underground and surface mining methods. The total number of projects planned for the near future or currently employing this technique include 17 commercial-scale operations and 12 major pilot scale operations. The South Texas uranium district and Wyoming sites represent all of the commercial-scale projects and most of the pilot-scale facilities. In 1980, it is estimated that approximately 10 percent of the total U.S. production of uranium was obtained from in situ mining methods. In contrast, in 1975, less than 1 percent of the total domestic uranium production was attributed to in situ mining methods.

This increasing level of activity in uranium production by in situ leaching methods has been accompanied by numerous research studies, primarily investigating technical aspects of production. Areas of investigation have generally focused on leaching chemistry, well field pattern design, solution flow characteristics, and extraction plant metallurgy.

Economics of uranium in situ leach mining have also been addressed at a broad level in public literature, and comparative cost estimates of this method and conventional mining techniques have been made. A more detailed assessment of costs, however, has been largely unavailable.

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[4] Mathematician, Twin Cities Research Center, Bureau of Mines, Minneapolis, Minn.

This report summarizes the results of an in-depth analysis of in situ uranium production costs by employing a process engineering approach. This approach disaggregates the in situ production process and analyzes each component in terms of its requirements and associated costs.

The conduct of this analysis effort has required that both a cost analysis procedure and a cost data base be generated. The cost procedure developed, termed a cost model, has been designed to offer the user maximum flexibility in specifying site conditions. The cost procedure, or cost model, accepts this site information and subsequently sizes both well field and extraction plant, determines equipment and manpower requirements, and assigns an appropriate cost from the model data base. This entire procedure has been computerized.

In this method, total capital and operating costs are generated for the life of the project. Costs are developed for all project activities from the point in time that a decision is made to commence pilot-scale operations through production termination and site reclamation. These costs are subsequently entered into a discounted cash flow analysis which solves for the production cost per pound subject to the rate of return identified by the user input data. Conversely, the model can solve for the rate of return on equity to be realized from a project for a specified sales price per pound.

The cost model developed under this research effort has been applied to typical in situ mining situations encountered in both Texas and Wyoming. Sensitivity tests were also conducted to identify the degree of cost influence exerted by incremental changes in key project parameters.

COST MODEL FEATURES

The cost analysis procedure and cost data base from this research effort have been developed over a period of approximately 1 year. Activities included visiting nine operating projects, making phone contact with many other operators, soliciting cost data from manufacturers and vendors, and drawing upon project team operating experience and in-house data.

The cost model that has evolved from this design framework contains the following features:

1. Develops detailed costs (both capital and operating) and requirements for any user-specified project condition for the life of the project.

2. Solves for minimum required sales revenue per pound of U_3O_8 (production cost) or rate of return on equity.

3. Contains regionalized data base for both Texas and Wyoming site applications.

4. Allows for cost analysis applications when only minimal information is known, as well as for cases in which detailed project data are available.

5. Accepts and accounts for either static or dynamic site conditions throughout project life.

6. Accepts user-specified capitalization structure options.

Each of these points is further explained in the following discussion.

1. Develops detailed costs.--Beginning with the point in time when a decision is made to start a pilot plant facility, all capital and operating costs are estimated by this model. The categories of costs generated are listed below:

Capital Costs

Process equipment
Equipment installation and/or site improvement
Building
Initial well field
Permitting
Pilot plant
Restoration system
Engineering and/or project management
Fixed capital
Contingency
Deferred capital

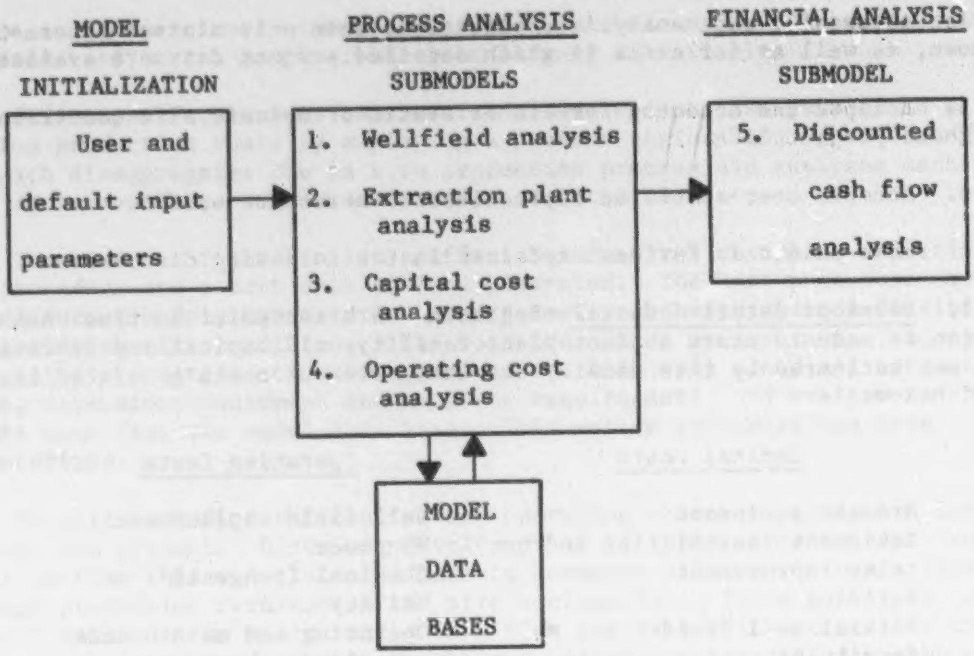
Operating Costs

Well field replacement
Manpower
Chemical (reagent)
Utility
Operating and maintenance supplies
Makeup water
General and administrative

Operating costs listed above are considered direct operating costs. Other noncash costs, such as depreciation and depletion, are also calculated in the discounted cash flow analysis (DCF). Royalities and local taxes are likewise estimated in the DCF analysis. The costs presented above represent those categories computed by submodels dealing with capital and operating cost estimates which all feed into the DCF analysis.

The process analysis component of the model essentially sizes the project. The basic user-specified site conditions are translated into requirements for extraction plant size (gallons per minute feed rate) and well field size (number of patterns and well fields) to meet the user-specified production level. Manpower, horsepower, and well field replacements are also computed.

These requirement calculations serve as the basis for appropriate cost assignments from the model data base. A simplified overview of the cost model process is presented on the following page:


MODEL INITIALIZATION
PROCESS ANALYSIS SUBMODELS
FINANCIAL ANALYSIS SUBMODEL
User and default input parameters
1. Wellfield analysis
2. Extraction plant analysis
3. Capital cost analysis
4. Operating cost analysis
5. Discounted cash flow analysis
MODEL DATA BASES

Submodels 1 and 2 above develop requirements and costs for the respective categories listed. Submodels 3 and 4 take both capital and operating cost components of the well field and extraction plant and generate total costs for each of these categories. The DCF analysis of submodel 5 solves for the sales revenue per pound of U_3O_8 or the rate of return on equity. Each of these submodels is further described later in this section.

2. Solves for minimum required sales revenue per pound of U_3O_8 or rate of return on equity.--The discounted cash flow analysis submodel provide the mechanism for either of the above solution options. Solving for one of these options requires knowledge of the other as a model input. The DCF analysis uses a profit and loss statement structure for financial analysis of the project.

A user solving for sales revenue per pound of U_3O_8 is interested in determining the minimum sales price that is adequate to cover all operating costs and capital recovery expenses and to provide a specified rate of return on equity invested in the project. In this situation, sales revenue per pound assumes an interpretation of economic cost of production.

A user solving for rate of return on equity will be employing a known market price for U_3O_8 as input to the model. This situation will be testing the viability of a project in terms of its rate of return yield at the anticipated market value.

Either of the solution options can provide valuable planning information for property screening or for testing alternative production levels or other project design factors for a given ore body configuration.

3. Contains Regionalized Data Bases.--The distinction between costs incurred in Texas and Wyoming project sites has been incorporated into the data base of the cost model. The primary variations recognized by the data base account for equipment price differences for similar equipment as well as for equipment and/or process system preferences typically associated with each region.

A summary of the key data base distinctions for Texas and Wyoming sites follows:

Extraction Process Equipment

1. Defines ion-exchange system typically employed.

2. Allows user four options (Upflow fixed bed, Downflow, Upflow Porter, Upflow USBM).

3. Defines cost differences for each ion-exchange system in Texas versus Wyoming for each equipment item for three plant sizes (400, 1,000, and 2,000-gpm plant feed rates).

Well Field Equipment

1. Defines drilling and casing cost differences for each region for three depth categories.

2. Defines surface piping cost differentials for insulated piping (Wyoming) versus standard PVC piping.

Capital Costs

1. Defines schedule of expenditures for each capital item according to Texas versus Wyoming site location.

2. Defines permitting cost differentials and time involved according to region.

3. Restoration system selection and therefore cost is based on region (deep well for Texas versus evaporation pond for Wyoming).

Operating Costs

1. Incorporates differentials in chemical reagent costs between regions.

2. Incorporates preferences for leaching solution between regions (ammonium carbonate-bicarbonate for Texas, sodium carbonate-bicarbonate for Wyoming).

3. Includes differences in power and labor costs between regions.

4. Allows for cost analysis applications under a wide range of information availability conditions.--To accommodate the broadest possible applications, the input structure of the cost model has been organized into three categories:

1. Required input.
2. Optional input (default values).
3. Calculation override input.

These three categories which appear in tabular form at the end of this paper, represent the range of information availability regarding a uranium in situ leach mining project. Category 1 contains the basic input parameters that must be known about a project in order to initiate a model run. There are 13 input parameters in category 1, organized according to physical, operating, and financial characteristics. Examples of these parameters are depth, grade, and ore thickness. There is also a series of cost update factors which may be used for cost base years other than 1980.

Category 2 input includes more detailed characteristics of the project which may not always be known. These parameters are assigned default values in most cases according to the site location. These default values will be used by the model calculation procedures unless the user specifies another value in the input sequence. Examples of parameters in this category include leach solution, pattern, type, and solution grade.

Category 3 includes those cost and requirement parameters that are calculated by the model. If information is available, however, on the specific costs of one or more of the parameters included in this listing, the model user may input the value when initiating the model run. This procedure will then negate any model calculation required for the subject parameter and instead use the value established by the user input.

5. Accepts and accounts for either static or dynamic site conditions throughout the project life.--This feature relates to the ore deposit geometry, chemistry, well field design, and anticipated flow rates and solution grades. The model user is given the option of specifying constant conditions throughout the project life for 10 parameters or of varying the conditions for each succeeding set of well field patterns.

When changing site conditions is more appropriate than using average values for the selected input parameters, the model user may specify changing values in terms of absolute or percentage values. To further demonstrate this option, the following example of ore body depth changes over the life of the project is presented:

Ore body data:	
Depth of well field 1............	400 ft.
Expected change..................	2-pct depth increase for each succeeding well field.
Input requirement structure: Depth	400,0.02.

This input data reflects an initial well field depth of 400 feet with an increasing depth for well field 2 of 408 feet. Succeeding well fields would be 416 feet, 424 feet, 433 feet, etc. If specific changes in depth are known in terms of absolute values, they may be input as follows:

Depth = 400, 420, 480, 410

In this case, the input indicates that well field 1 is 400 feet deep, well field 2 is 420 feet deep, and well fields 3 and 4 are 480 feet and 410 feet, respectively.

Values for up to 30 well fields may be input in this manner. Other parameter inputs that may be varied in a similar fashion include

Depth of deposit
Thickness of deposit
Ore grade
Solution grade
Well field pattern type
Injection to production well ratio
Injection to production well spacing
Production well flow rate
Monitor well fraction of total wells
Recovery or percent of contained reserves

6. Accepts user-specified capitalization structure options.--The model user may indicate any debt-equity capitalization structure for the project being analyzed. Further, the length of loan payback as well as the debt servicing rate may also be established by user input.

This flexibility offers obvious advantages for testing the effects of alternative project financing arrangements and the sensitivity of rate of return or cost per pound of U_3O_8 to variations in any of the capitalization parameters.

All of the above-mentioned features are indicative of a costing tool which has been designed for maximum user flexibility.

The model will facilitate rapid sensitivity testing of the effect of the project parameter changes on cost results for a specific site. It will serve a useful function in preliminary screening of properties for economic viability. Alternative well field designs or extraction plant systems may likewise be quickly examined.

The model is not designed to predict key project parameters such as production well flow rate or solution grade based on permeability, depth, or other influencing factors. Many parameters, however, have been assigned default values based on regional location which may be used or overriden by model users.

The primary value of this model is its ability to quickly translate any user-dictated values for such parameters into overall project cost and design implications.

COST MODEL INPUT STRUCTURE

The input structure of this cost model is organized into three distinct categories: (1) required input, (2) default assigned input, and (3) calculation override input.

In progression from 1 to 3, these categories represent increasing levels of information availability regarding project conditions.

Category 1 represents the minimum amount of information required to initiate a model run. Category 2 includes default values, or expected values for a number of parameters that will be applied in the costing procedure unless changed by the user input. Category 3 contains a parameter listing for major cost and requirement results that are calculated during a model run. Should the model user have specific cost information for a particular cost category, that value may be input--thereby negating all model calculation relating to that category and instead applying the input value.

Each of the three categories of input is further subdivided into physical, operating, and financial parameter groupings. Category 1, however, has a fourth subdivision termed a control parameter. This provides the user with the option of designating the specific output tables desired.

A further model input option allows the user to designate only a single project life value or multiple values for any of 10 parameters describing ore body and well field conditions. This input option allows the more technically informed model user to dictate any anticipated changing conditions as mining advances through the ore body.

The parameters included in each of the three input categories are tabulated at the end of this paper. The corresponding model acronym for each parameter is also presented. Parameters for which multiple values or percentage change values may be input are designated by an asterisk. Under the category 2 parameters, the default values assigned by the model are also listed for each situation in which they would be applied.

Most values in category 2 are automatically assigned based on the regional location of the project (Texas or Wyoming). Some, however, are applied in both cases or are based on ore body characteristics. The values used as default assumptions are based on practices or situations most commonly observed or reported for Texas or Wyoming projects.

It must again be stressed that any of these default values may vary from site to site, and the model user may respond to such variance by overriding these default values as project conditions dictate.

The purpose of including such values is to enable broad-level analysis to be conducted in the absence of detailed technical project information. The results of applying the cost model when using the default assumed values must therefore be interpreted to have wider degrees of uncertainty--unless, of course, the default values coincide with perceived project conditions.

SUMMARY

The Bureau of Mines, in conjunction with the NUS Corp., has developed a computerized cost model for uranium in situ leach mining. The model is structured to handle a wide range of user options, so that the novice or more experienced individual can utilize the system. Currently the Bureau is looking for cooperators to use and verify the model so that its effectiveness, from an industry point of view, can be determined. Future work will be directed towards expanding the model's capabilities to other commodities, so that a wide range of in situ mining costs can be obtained.

COST MODEL INPUT STRUCTURE

Category 1.--Required input parameters

	Acronym
Physical parameters:	
Depth of deposit (ft) (May be stated as an initial value and percentage increase or decrease for each new well field. Conversely, absolute footage values for each subsequent well field may be used.)	DEPT*
Thickness of deposit (ft) (Initial footage value and same as DEPT.)	OTH*
Ore grade (Initial percentage value and same as DEPT.)	OGRA*
Location of property (Texas or Wyoming)	LOC
Operating parameters:	
Annual production of U_3O_8 (lb)	ANNP
Productive life (years)	PDTL
Financial parameters:	
Sales revenue per pound (dollars)	SRV
Rate of return of equity (pct)	ROR
Debt-financed portion (pct)	DEBT
Debt-servicing rate (pct)	DSR
Project start year (calendar year)	PSY
Cost base year (calendar year)	CBYR
Cost Update Factors:	
Extraction plant	EXPF
Drill and case wells	DCWF
Well field equipment	WELF
Mobil equipment	MOBF
Chemical costs	CHCF
Power costs	PWRF
Manpower costs	MPCF
Restoration equipment	RESF
Control parameters: Print tables	PRNT

*Same option as DEPT offers.

Category 2.--Default assigned input parameters

	Acronym	Value
Physical parameter:		
Recovery (decimal)	REC*	0.7.
Operating parameters:		
Leachant	LEAC	Texas - ammonia. Wyoming - sodium.
Oxidizer	OXID	<200 ft depth - H_2O_2. ≥200 ft depth - O_2.
Solution grade	SG*	Texas - 50 ppm. Wyoming - 80 ppm.
Extraction plant process	EXPP	Texas--upflow continuous USBM. Wyoming - downflow. Other options (Texas and Wyoming): Upflow fixed bed; upflow continuous - Porter.
Well field pattern type	PTYP*	Texas - 5 spot. Wyoming - 7 spot. Other option (Texas and Wyoming): Line drive.
Injection-production well ratio.	INPR*	Texas - 2 to 1. Wyoming - 3 to 1.
Injection-production well spacing.	DIST*	Texas - 50 ft. Wyoming - 40 ft.
Production well flow rate	PWFR*	Texas - 20 gpm. Wyoming - 10 gpm.
Operating schedule (days per year).	OSCH	Texas - 350. Wyoming - 340.
Monitor well fraction of total wells.	MWPT*	0.10
Financial parameters:		
Overhead charge	OVHD	35 pct.
General and administrative	GNA	5 pct.
Royalty charge:		
Percent of selling price	ROYP	0
Dollar per pound of charge	ROYC	0
State taxes	STXP	0
Percent of selling price	STXC	0
Federal income tax	FIT	46 pct.
Miscellaneous operating expense.	MSOE	$0 per pound.
Acquisition cost	ACQ	0
Capital cost contingency factor.	CCON	10 pct.

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*Multiple values or percent change values may be input.

Category 3.--Calculation override input parameters

	Acronym
Physical parameter: None	
Operating parameters:	
Hourly labor requirements	HLAB
Salaried personnel requirements	SALP
Preproduction development time	DTIM
Extraction plant size	EPS
Pattern life	PTNL
Financial parameters:	
Process equipment cost	PEC
Installation and site improvement cost	ISI
Building costs	PBC
Well field costs (allow 1 cost per well field input)	TWC
Restoration system cost	RESC
Permitting costs	PERC
Pilot plant costs	PPC
Site reclamation costs	SRC
Engineering project management costs (percent of fixed capital)	EPJM
Direct operating costs	DOC

BRANCHED BOREHOLES FOR IN SITU LEACH MINING

by

William C. Larson,[1] Don W. Dareing,[2] Ed T. Wood,[3]
and Don H. Davidson[4]

ABSTRACT

In situ leach mining now offers a third viable option along with open pit and underground mining methods for the extraction of mineral values. Multiple-branch wellbore and horizontal holes, when applied to deep-lying ore bodies, have the potential of significantly reducing well costs by reducing total footage drilled per acre of well pattern. In addition, horizontal holes may increase sweep efficiency. Well completion is a major problem, and the wells must be cased to contain leach solutions within the underground portion of the production loop. Several drilling and completion concepts are given and evaluated. The results show that there is economic incentive to further develop these concepts for field application to ore bodies greater than 1500 feet. Multiple-branch concepts can reduce well costs by as much as 30 percent when applied to 2,000 foot ore body depths.

INTRODUCTION

The process of in situ leach mining provides an opportunity to develop resources that are currently uneconomical to mine using conventional surface or underground mining techniques. In fact, in situ mining has been demonstrated to be a third option when considering the economics of an ore body, particularly uranium. In situ mining is being commercially practiced in relatively shallow deposits (200 to 600 ft).[5] However, mineral deposits of uranium, nickel, copper, molybdenum, and manganese are known to exist at much greater depths. Recovery of these deep-lying minerals (>1,500 feet) by in situ mining methods is dependent on the availability of relatively low-cost drilling and completion techniques.

Through its in-house and contract research program, the Bureau of Mines evaluated nontypical wellbore configurations such as branch holes (fig. 1) and drainholes (fig. 2) for in situ mining of deep deposits. Initial results by

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[2] Engineering associate, Maurer Engineering, Inc., Houston, Tex.

[3] Engineer, Completion Technology Co., Houston, Tex.

[4] Assistant project manager, Resources Development and Operation, TRW Inc., McLean, Va.

[5] Larson, W. C. Uranium In Situ Leach mining in the United States. BuMines IC 8777, 1978, 68 pp.

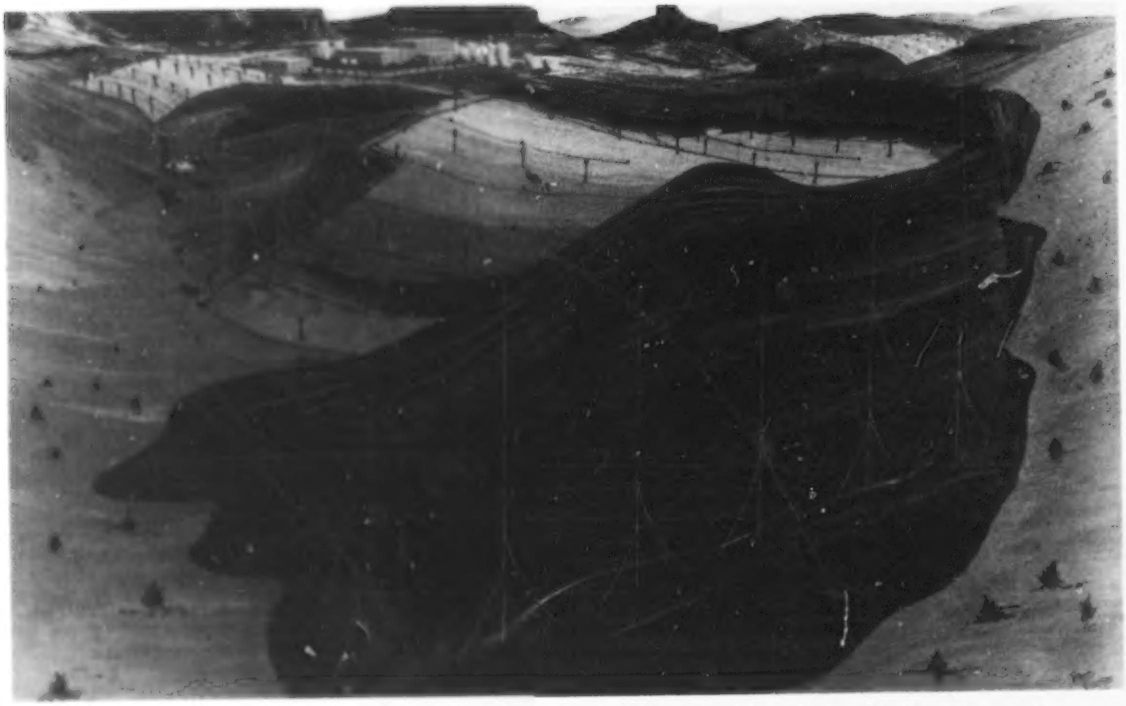

FIGURE 1. - Conceptual production scheme using branched boreholes for in situ mining.

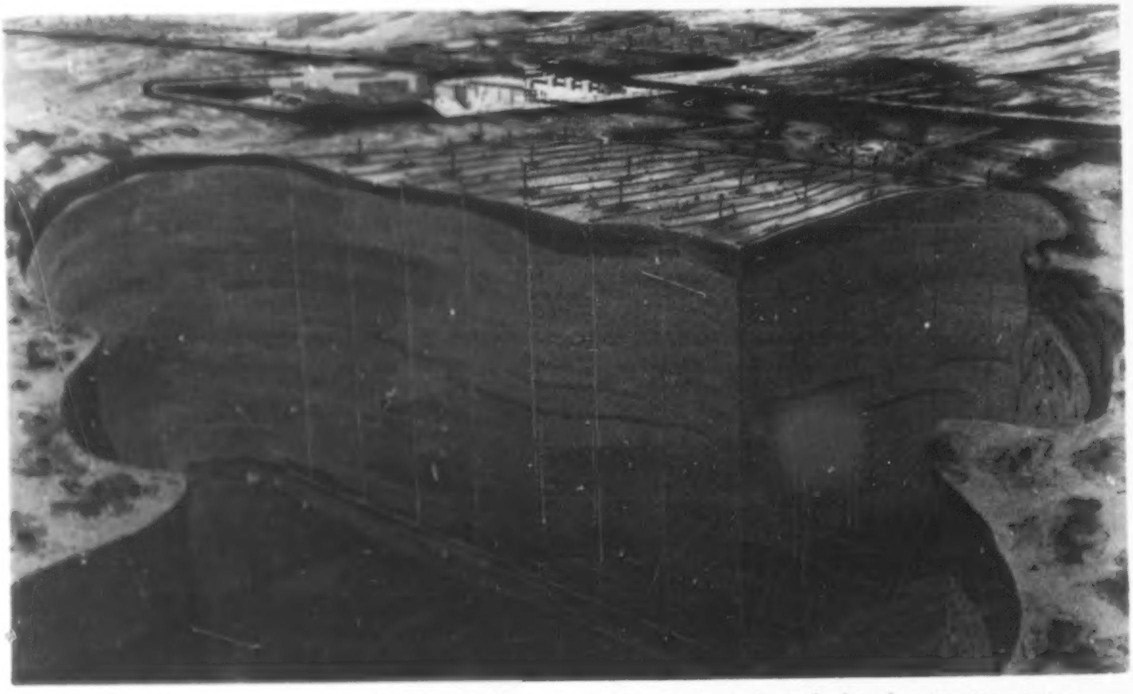

FIGURE 2. - Conceptual production scheme using horizontal drain holes for in situ mining.

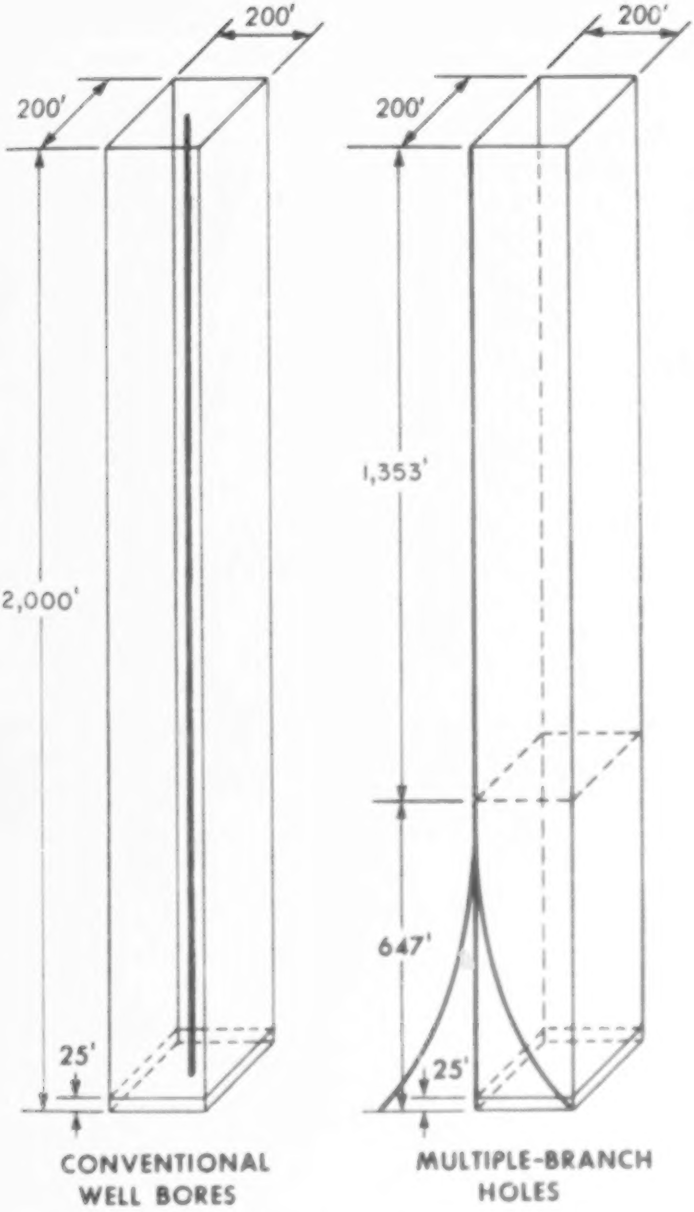

FIGURE 3. - Placement of conventional wellbores and multiple-branched boreholes.

the Bureau[6] showed that there are a number of advantages when using branch wellbore technology, such as increasing the efficiency of fluid sweep over a well pattern; reducing the pressure gradient in the well pattern, thus achieving higher flow rates per well; and finally reducing the total footage of overburden that must be drilled.

Conventional wellbores are usually placed in a five-spot pattern (fig. 3), where the center well is a producer and the four corner wells are injectors. This well pattern would sweep a given area, say 200 by 200 feet. In a broad field development program, each corner injector well would be shared with three other adjacent sweep areas so that the total well cost for a given sweep area is the cost of one producer and one injector.

Multiple-branch holes can be arranged to penetrate the ore body in a five-spot pattern with fewer wellheads at the surface. One approach is three injectors out of one vertical wellbore and three producers out of a separate wellbore as shown in figure 3; completion of these types of injectors and producers will be discussed later in the report. Fluid flow through the ore body would be the same for both conventional and multiple-branch flow cases. Well costs for one sweep area would be one-third the cost of a triple branch injector plus one-third the cost of a triple-branch producer.

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[6] Larson, W. C., and R. J. Morrell. In Situ Leach Mining Method Using Branched Single Well for Input and Output. U.S. Pat. 4,222,611, Sept. 16, 1980, 4 pp.
Morrell, R. J., W. C. Larson, and R. D. Schmidt, Method of In Situ Mining. U.S. Pat. 4,249,777, Feb. 10, 1981, 4 pp.

This project conducted by Maurer Engineering Inc., and sponsored by the Bureau of Mines Twin Cities Research Center, was designed to assess whether petroleum engineering technology related to drilling and completing branch and horizontal holes could be adapted to in situ mining to either enhance mineral recovery or reduce capital and operating costs. The incentive for evaluating this technology evolves from current interest in extending in situ mining to depths of several thousand feet below the surface. Since the total allowable subsurface investment is fixed within a narrow range, the number of well patterns in operation at any one time will have to decrease as mining depths increase, unless techniques are developed to reduce unit subsurface costs.

DRILLING AND COMPLETION CONCEPTS FOR IN SITU MINING

Branched wells have been drilled in the past, but none have been cased to allow leakproof and pressuretight communication throughout the wellbore. These, of course, are operational requirements for in situ leach mining. This section describes several concepts, generated by the project team, for drilling and completing nontypical wellbores.

The incentive for applying new wellbore types to 500-foot ore bodies is marginal because a conventional vertical hole that satisfies the requirements of in situ mining can be drilled and completed at a relatively low cost. It appears that the new technology would not be applicable above 1,500 feet. Therefore, the following concepts are directed primarily at ore bodies located at depths of 2,000 feet and beyond. At these depths, the economics for using advanced drilling and completing technology look much more favorable. The following sections describe three concepts for drilling and completing branched boreholes for in situ mining.

Triple Branch Out of 13-3/8-Inch Casing

A triple-branch well consists of a vertical protection casing with three branches extending into the ore body. A series of parallel rows of producers and injectors can be used to develop a five-spot sweep area with less total footage drilled than with conventional wells.

This drilling and completion scheme consists of running large-diameter, low-grade steel protection casing containing a drilling template (fig. 4), then drilling and completing three branch wells out through the bottom of the casing as illustrated in figure 5. The bottom joint of casing contains the drilling guide and an internal indexing dog to allow for positive entry into the three-branch whipstock.

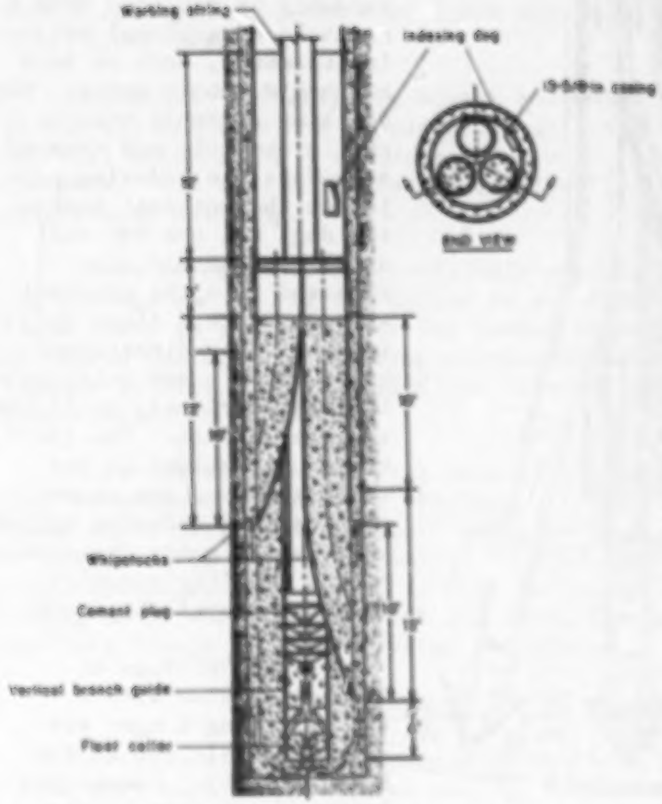
Working string
Indexing dog
13-3/8-in casing
End view
Whipstocks
Cement plug
Vertical branch guide
Float collar

FIGURE 4. - Cementing the 13-3/8-inch casing with drilling guide as the first step in branch well drilling.

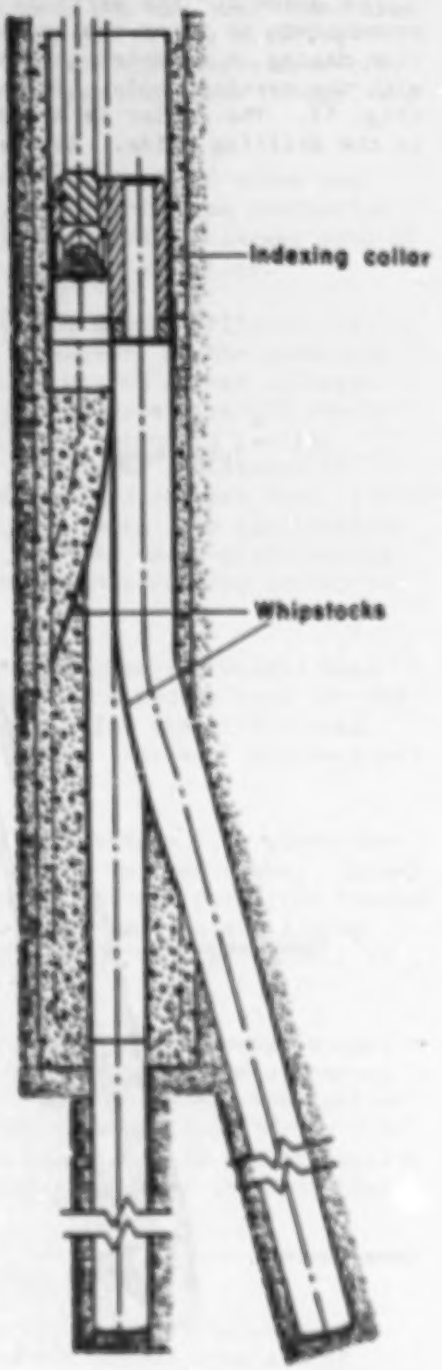
Indexing collar
Whipstocks

FIGURE 5. - Drilling and completing branched boreholes through the bottom of the casing.

A guide for the verticle branch contains a float collar and seal bore to accommodate an inner tubing string for cementing purposes. Once the protection casing is cemented in place, the three branches are drilled starting with the vertical hole. In each case, an indexing collar is run on the bit (fig. 5). The collar is keyed to orient itself with the internal indexing dog in the drilling guide. As the indexing collar lands on the dog, the bit is released and enters the appropriate branch. After the hole is drilled, the guide collar is retrieved by pulling the bit out of the hole.

The indexing collar then is rekeyed to index the drilling assembly into the second hole. Since the second and third branches are directionally drilled, the conventional drilling assembly is replaced with a downhole directional drilling assembly, such as bent sub and downhole motor. The drilling assembly then is run in the hole and rotated to locate the indexing collar on the internal indexing dog, and the bit and drilling assembly are released into the appropriate whipstock. After drilling the first directional hole, the bit and guide collar are retrieved, as in the vertical branch. The third branch is drilled in the same manner as the second; that is, the indexing collar is keyed to guide the directional drilling assembly into the proper whipstock.

Triple strings of fiberglass pipe with a triple tubing hanger are simultaneously run in the hole (fig. 6). Cement baskets are attached to the shoe of each casing string to prevent the fiberglass

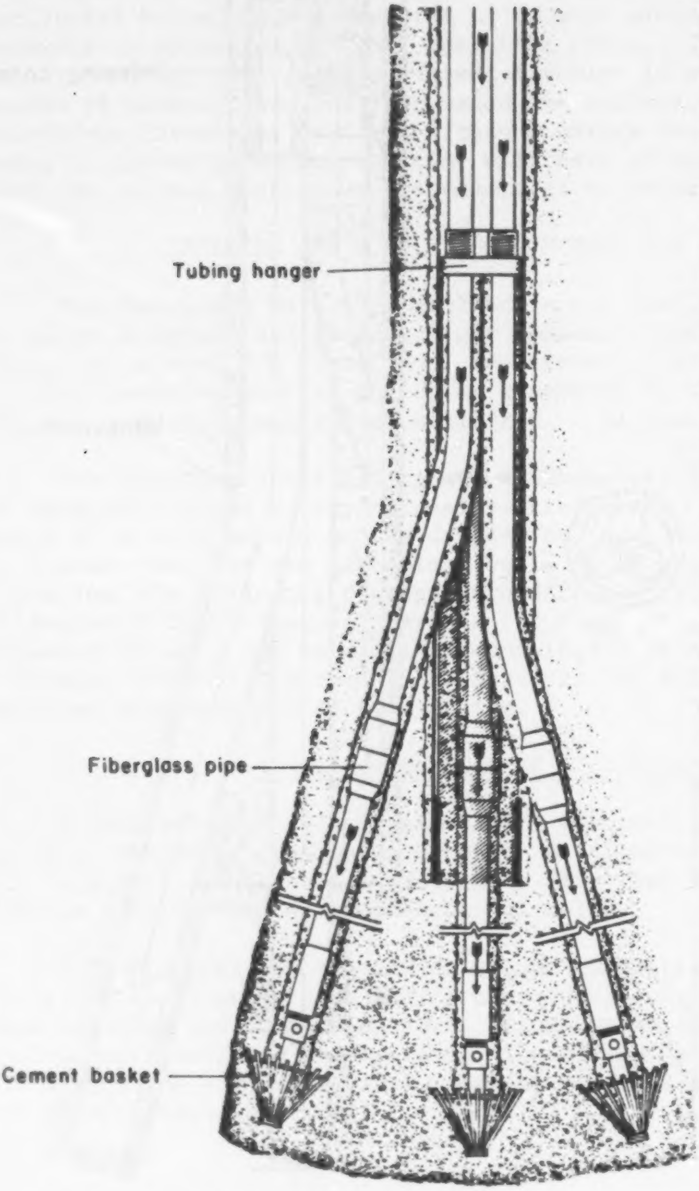

FIGURE 6. - Schematic diagram of branched boreholes with casing cemented into place.

pipe from floating as heavy cement is circulated into the annulus. The three tubing strings are oriented into the branches by the top of the drilling guide. Once in place, the three casing strings are simultaneously cemented. The triple tubing hanger is set, cement is reversed out above the hanger, and the three branches are perforated.

Production from a triple-branch well requires a fluid head above the branch point. A submersible pump would be set in this area from production tubing. The 13-3/8-inch casing allows more space for either a larger pump of improvements in pump designs, and we see this as a major advantage.

Setting 13-3/8-inch casing at relatively shallow depths utilizes standard oilfield drilling and cementing techniques. However, as the technique is applied to deeper objectives, care must be taken not to exceed collapse resistance of the casing. As a rule of thumb, the collapse strength should be greater than external hydrostatic forces acting on evacuated casing. For example, 13-3/8-inch, 54.5-lb/ft casing has a collapse resistance of 1,130 psi. Assuming a formation pressure gradient of 0.5 psi per foot, 13-3/8-inch, J-55-grade casing could be safely set at 2,260 feet. For applications below this depth, stronger casing must be used; 72-lb/ft, N-80-grade casing with a collapse resistance of 2,670 psi could extend safe setting depths to below 5,000 feet.

Techniques for drilling branch boreholes are low risk and widely used in the oil industry. A conventional drilling assembly would be used for the vertical branch, and a downhole motor with a bent sub for the directional branches. The rate of deviation (5°/100 ft) is within limits of conventional directional drilling.

There is not sufficient oilfield experience with triple fiberglass tubing strings set at shallow depths to accurately assess related risks. Actual field tests in shallow wells are needed to determine failure rate from tangled or kinked tubing. Triple-completion equipment such as packers and tubing hangers is available from oilfield service companies but in less variety than dual-completion hardware.

A review of relative risk related to the proposed branch design suggests that formation integrity is critical. Branch wells should not be attempted in areas where caving and washouts are a serious drilling problem. Application of branch wells should be limited to well patterns with smaller (50- or 100-foot) spacings. The same 5°/100 ft deviation rate would allow the protective casing to be set closer to the ore body, resulting in shorter branches that are less likely to cave in prior to casing.

Triple Branch Out of 9-5/8-Inch Casing

Directional drilling techniques can also be applied to triple-branch wells with smaller protection casing. The completion scheme consists of running 9-5/8-inch casing with an internal indexing dog to orient whipstocks toward windows in the protection casing. Branches are drilled and a tubing guide is installed to direct fiberglass casing into the branches. A series of five-spot patterns is developed by alternative parallel rows of producers and injectors.

Branch wells with smaller protection casing offer several advantages over the 13-3/8-inch concept. Small drilling rigs can be used to drill 12-1/4-inch holes and set 9-5/8-inch casing. Less pump volume is needed to circulate cuttings, and less rig power is needed to set casing. Potential for extending applications to greater depths is also greater with small casing, effectively increasing the value of experience gained at shallow depths.

For completion of branch wells with smaller protection casing (9-5/8-inch), the following approach is suggested. Protection casing is set an appropriate distance above the ore body. The bottom joint contains a prefabricated float assembly and seal bore for an inner string or stab in cementing, and an internal indexing dog to positively locate each branch (fig. 7). Fiberglass-filled (or other material) windows are provided as easily penetrated exit points for the two directional branches.

After the protection casing is cemented in place, the vertical 6-inch branch is drilled, (fig. 8). A whipstock assembly then is run in the protection casing and rotated to seat on the internal indexing dog. When in place, the whipstock guides a directional drilling assembly through the premilled fiberglass window in the bottom joint of the protection casing (fig. 9). After the directional branch is drilled to the appropriate depth, the drilling assembly is pulled and the whipstock is retrieved with a whipstock-pulling assembly.

The whipstock assembly then is modified to guide the directional drilling assembly into the upper window in the protection casing. The whipstock is run in the hole with a running assembly and rotated to land on the internal indexing dog with the whipstock facing the second premilled window (fig. 10). The upper branch is drilled directionally to the appropriate depth.

A triple tubing guide then is installed using the internal indexing dog for proper orientation (fig. 11). Once in position, it will guide the three branch casing strings into appropriate holes.

A triple string of fiberglass pipe and a triple tubing hanger then are simultaneously run in the hole. Cement baskets are attached to the shoe of each casing string to prevent the fiberglass from floating as heavy cement is circulated into the annulus. The top of the tubing guide orients the three strings into the branches. Once in place, the three casings are simultaneously cemented and the tubing hanger is set. Cement is reversed out above the hanger; the three branches are perforated, and the well is ready for injection.

Setting 9-5/8-inch casing utilizes standard oilfield drilling and cementing techniques, and higher grade casing is available for applications down to 5,000 feet and deeper.

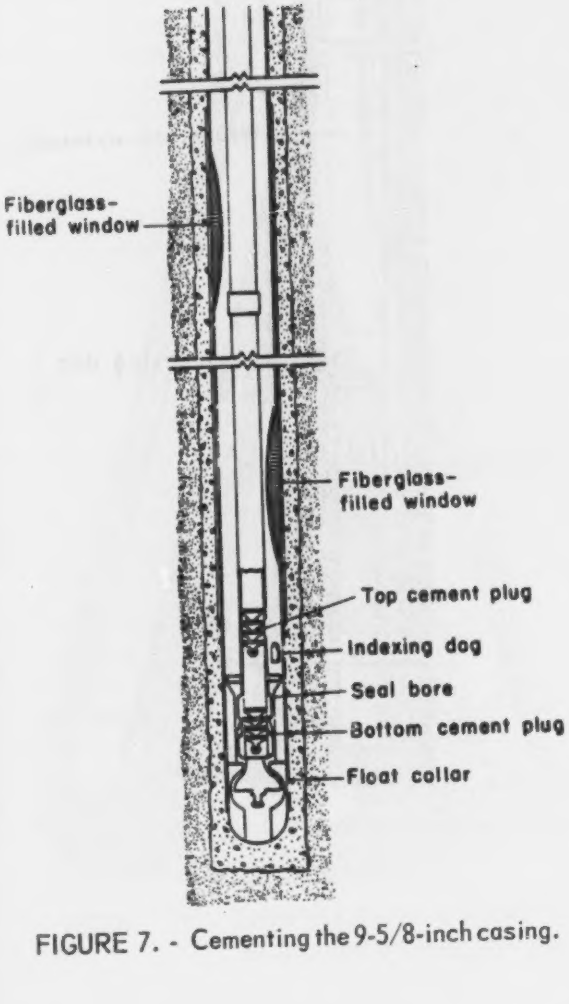
Fiberglass-filled window
Fiberglass-filled window
Top cement plug
Indexing dog
Seal bore
Bottom cement plug
Float collar

FIGURE 7. - Cementing the 9-5/8-inch casing.

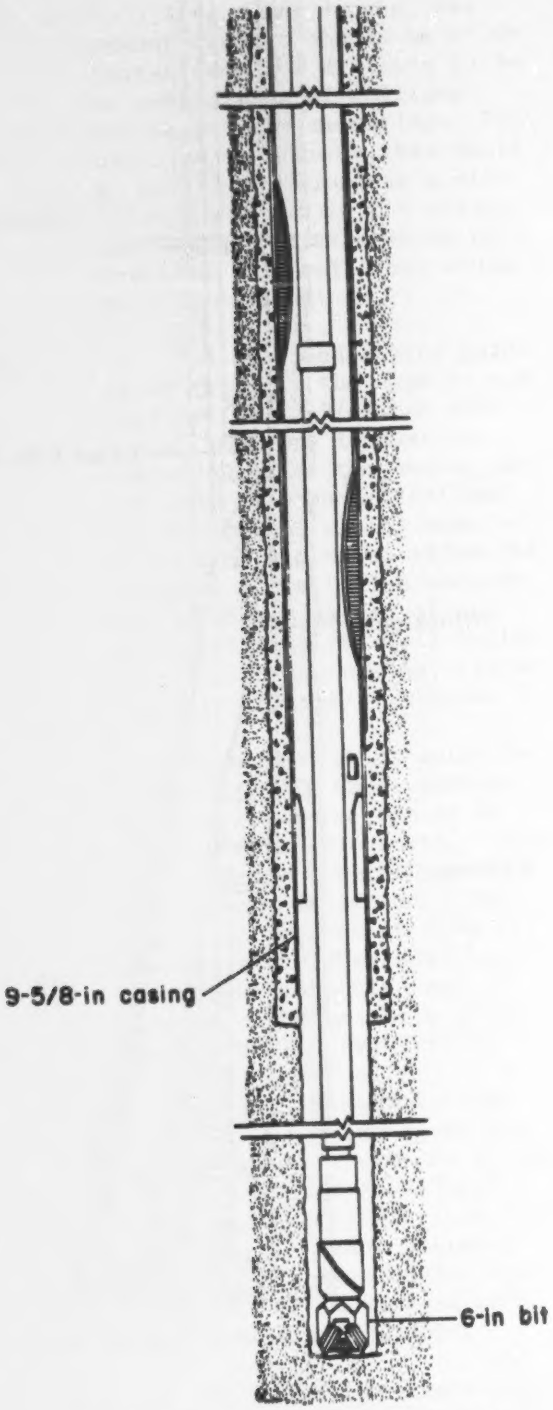
9-5/8-in casing
6-in bit

FIGURE 8. - Drilling the first branch.

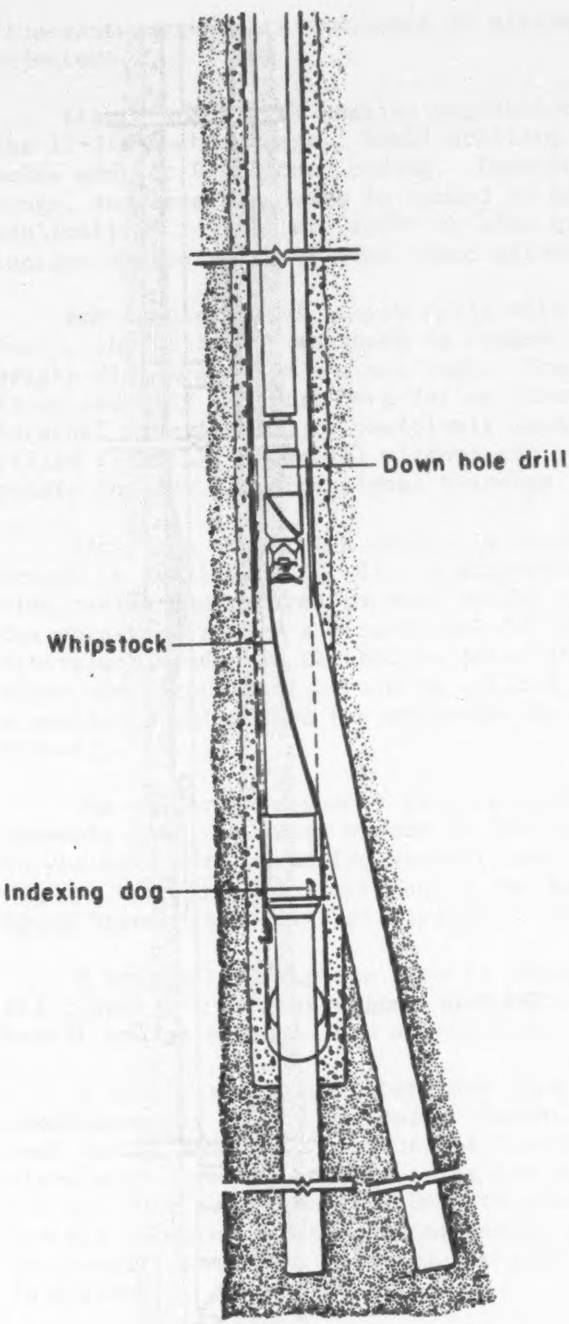
Down hole drill
Whipstock
Indexing dog

FIGURE 9. - Drilling the second branch.

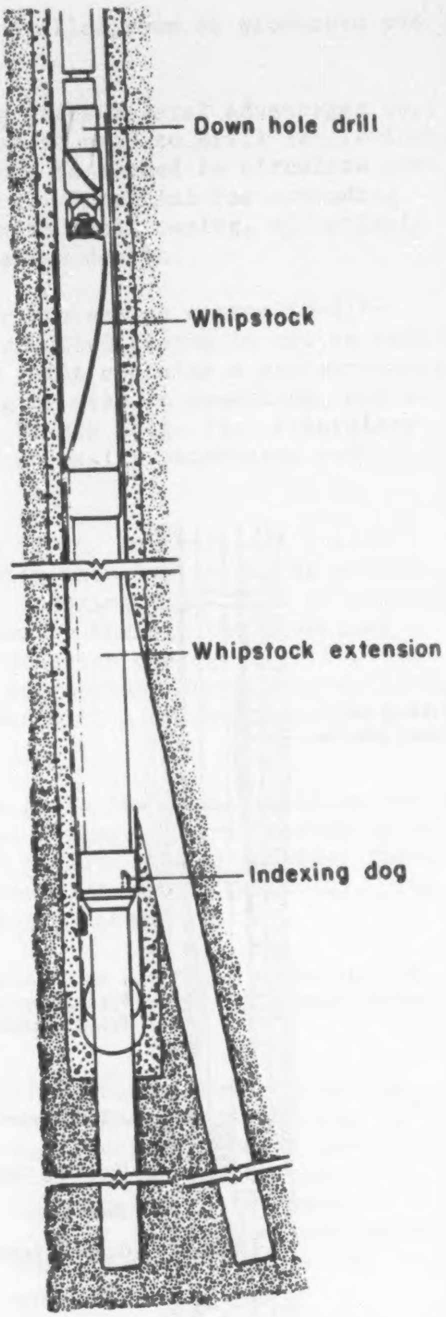
Down hole drill
Whipstock
Whipstock extension
Indexing dog

FIGURE 10. - Drilling the third branch.

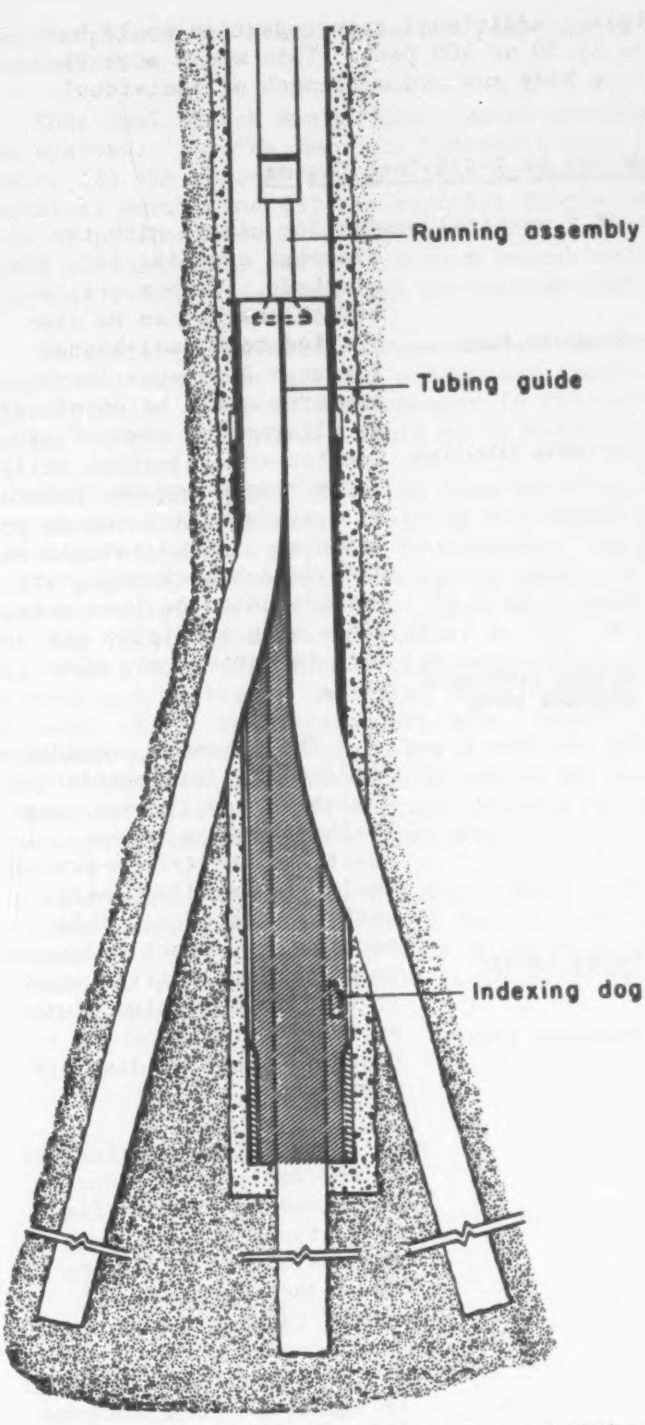

FIGURE 11. - Installing the triple tubing guide before installation of fiberglass casing.

At greater depths, the precut windows should be eliminated from the drilling guide to reduce risk of collapse during primary cementing. The drilling procedures then would be modified to include a mill to open windows in the casing. Additional rig time would be involved, and well cost would go up accordingly.

The proposed tubing guide is essentially the same as was used in the 13-3/8-inch concept. However, it must be installed after the casing is set and the branches drilled. The 9-5/8-inch casing must be in good condition to allow the guide to be installed because the outside diameter of the guide utilizes the full inside diameter of the casing, allowing only minimum clearance.

Once the tubing guide is installed, the three strings of fiberglass are run as in the 13-3/8-inch concept. This is the most critical phase of the branch completion. The open hole section of each of the three branches must have sufficient integrity not to cave or collapse while subsequent branches are drilled.

At this stage of completion, remaining risks are essentially the same as in the final stages of the 13-3/8-inch concept. The triple tubing hanger, cement baskets, and cementing procedures are identical to those used with the 13-3/8-inch concept.

Oilfield experience suggests that a dual-branch well would have considerably less completion problems than the triple. Additional risk reduction would be attained by reducing well spacing to 50 or 100 feet. This would move the protection casing closer to the ore body and reduce length of individual branches.

Double Branch Out of 9-5/8-Inch Casing

A dual-branch well consists of a vertical protection casing with two branches exending into the ore body. Either triple-branch design can be simplified to a dual-branch design. Five-spot leach patterns would be developed by alternating rows of production and injection wells or by completing one branch as a producer and one as an injector. Risks associated with dual-branch wells are considered to be less than those with a triple, but cost incentives are also less.

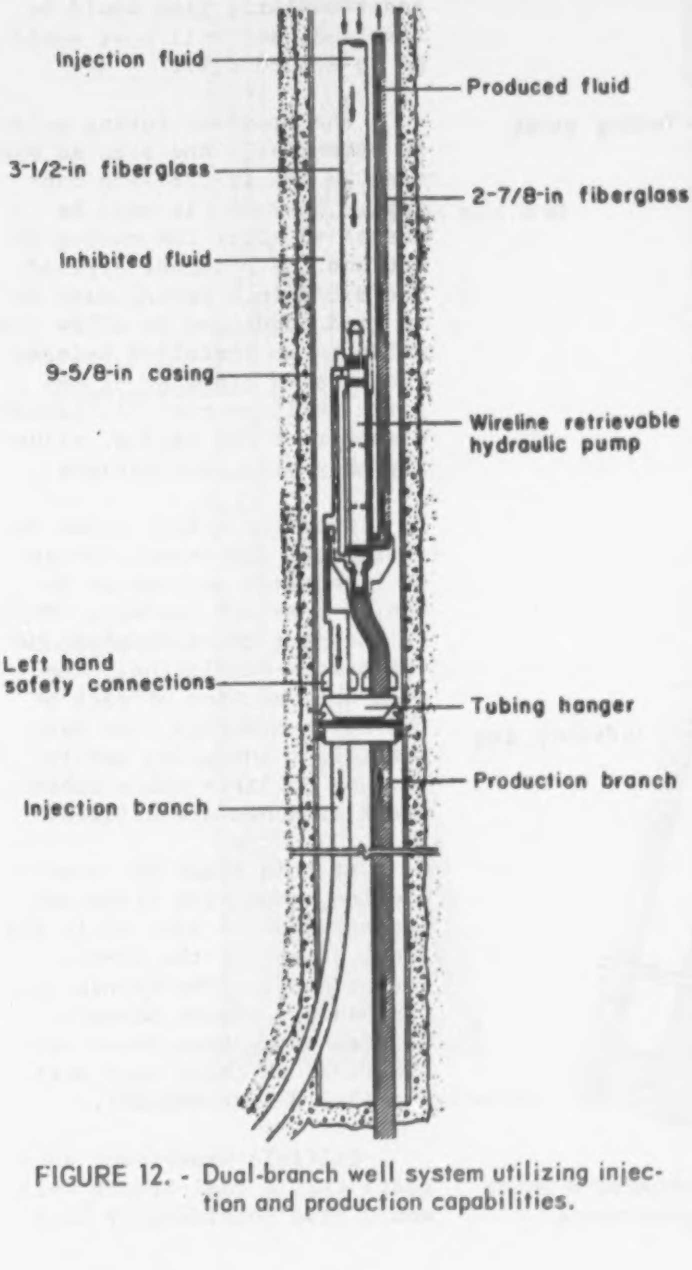
Injection fluid
Produced fluid
3-1/2-in fiberglass
2-7/8-in fiberglass
Inhibited fluid
9-5/8-in casing
Wireline retrievable hydraulic pump
Left hand safety connections
Tubing hanger
Production branch
Injection branch

FIGURE 12. - Dual-branch well system utilizing injection and production capabilities.

The proposed procedures for drilling and completing dual-branch wells are essentially the same as was described for triple-branch wells with small-diameter casing. Therefore, this section is primarily devoted to designing a lift system to utilize injection fluid as the power source for a downhole positive displacement pump.

One unique application of the dual-branch concept is to use one branch for injection and one branch for production, as in figure 12. The injection fluid is routed through a positive displacement downhole pump as the power fluid; it then is exhausted from the pump into the injection branch. Produced fluid is routed from the production branch into the pump and up the production string, as illustrated.

This dual branch design has several advantages over conventional production systems: (1) The downhole hydraulic pump is wireline retrievable for repair, (2) the scheme requires less surface plumbing and requires no downhole electrical power, and (3) the positive displacement pump maintains a constant ratio of produced fluid to injected fluid. For example, the system shown in figure 12 utilizes a dual 9-5/8-inch branch well with 3-1/2-inch fiberglass injection tubing and 2-7/8-inch production tubing.

In reality, operating parameters of the dual-branch system will be in a dynamic state. Injection and production pressures will vary with influence of adjacent wells and temporary changes in effective permeability. However, the critical ratio of produced fluid to injection fluid will remain constant.

Dual-branch wells apply the same drilling and completion procedures as were proposed for triples. Setting 9-5/8-inch protection casing at 1,350 feet uses common low-risk drilling procedures. Specialty equipment such as the drilling guide, whipstock, and tubing guide are conceptually the same as proposed for triple-branch wells. Risk associated with specialty drilling equipment for duals therefore is similar to that discussed for triples. However, a dual offers some significant risk reductions in that less time elapses between drilling and casing of the first branch--hence there is less chance of losing the hole. Also, completions are more common in oil wells, and a wide variety of packers and tubing hangers are available with experienced people to install them. Additional risk reductions can be gained by reducing well spacing to 50 or 100 feet, allowing the vertical protection casing to be set deeper, thus reducing arc lengths of the branches.

The pumping system in the dual-branch example is novel. A standard positive displacement pump would require metallurgical modifications to utilize corrosive leach solutions as power fluids. However, advangages such as wireline retrieval and fixed ratio of production and injection fluids warrant a further study.

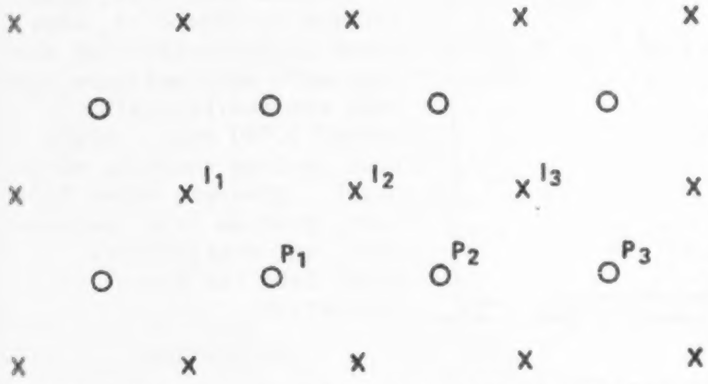

FIGURE 13. - Five-spot field development pattern.
x = injector; o = producer.

COMPARISON OF WELL COSTS PER SWEEP AREA

Consider that a broad mineral field is to be developed by a matrix of injector and producer wells drilled and completed in a five-spot pattern as shown in figure 13. At present, the matrix of injectors and producers comprises conventional vertical wells, and the well cost per sweep area is the total cost of one conventional injector well and conventional producer well.

If the mineral field is to be developed using triple-branch injector wells (I_1, I_2, I_3) and triple-branch producer wells (P_1, P_2, P_3), the well cost per sweep area is one-third the cost of a triple-branch injector well plus one-third the cost of a triple-branch producer well. This formula was used to generate the sweep area cost for both triple-branch cases (table 1).

TABLE 1. - Well cost per sweep area

Completion method	Ore body depth	
	2,000 ft	5,000 ft
Conventional....................................	$181,700	$384,160
Triple branch out of 13-3/8-in casing..........	122,603	235,634
Triple branch out of 9-5/8-in casing...........	125,924	214,106
Double branch out of 9-5/8-in casing...........	172,158	291,662

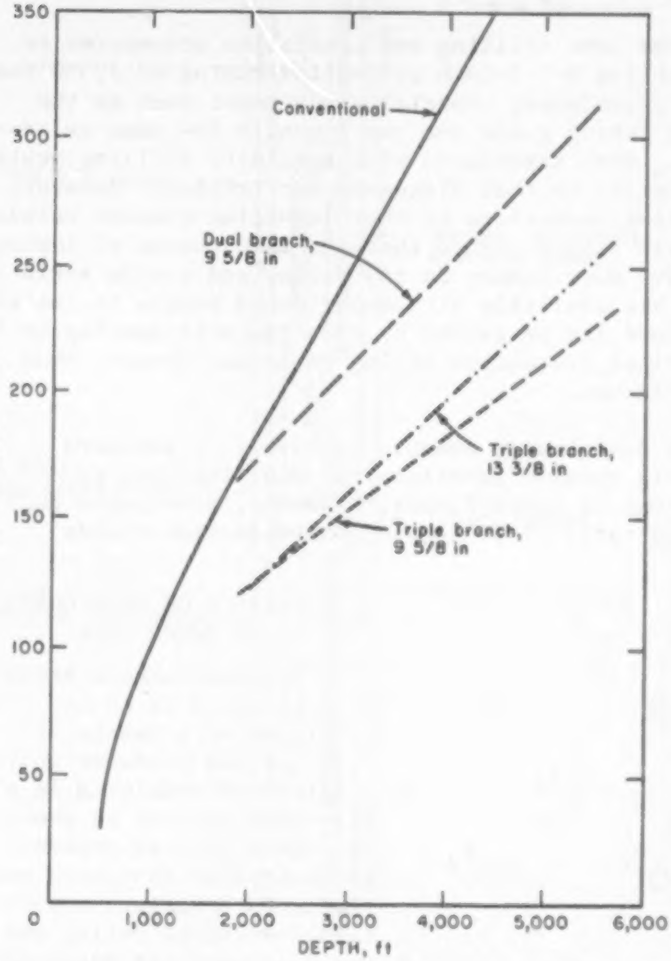

FIGURE 14. - Well cost per sweep area using branch boreholes for in situ mining.

The double-branch well is costed assuming one branch will be an injector and the other a producer. In this case, the cost of a double branch is the same as the well costs to sweep one area (200- by 200-foot sweep area). Drilling costs are site specific, and rig rates vary with demand. Also, distance relative to an active oilfield significantly changes the expense of equipment rental. For comparative purposes each well scheme is priced as though it would be drilled in the Houston area.

These cost data, also plotted in figure 14, show there is potential cost savings with each wellbore type when applied to depths beyond 1,500 feet. Also, cost savings increase with depth. However, other factors, such as risk, performance, and availability, enter into the overall evaluation.

CONCLUSIONS

Branch well drilling can reduce well costs when applied to mineral deposits deeper than 1,500 feet. The practical limit for number of branches drilled and completed from one vertical wellbore is three. The two drilling and completion concepts for triple-branch wells require development of specialized completion templates and guides. The logical first step in developing branch well completion equipment is to limit initial branch wells to include only two hole bottoms. Completion experience gained by developing templates and guides for dual-branch wells could be readily extended to expertise needed to complete three hole bottoms. Further studies will be required to develop the specialize equipment proposed for branch well completions and to determine risks related to their use.

GEOCHEMICAL KINETICS MODEL FOR IN SITU LEACH MINING

by

Robert D. Schmidt,[1] Steven E. Follin,[2] Kent A. Peterson,[3]
and Eric V. Level[4]

ABSTRACT

A computer model of in situ leaching chemical kinetics is presented as an analytic, predictive tool, useful in determining the leachability and productive potential of an undeveloped ore deposit and in the optimal design of an operating well field. Some examples of model application are presented. Special emphasis is placed on explaining model usage.

INTRODUCTION

Changes in leach site operating conditions (for example, injection well pumping rates) will affect the net productivity of a pattern owing to the complex interactions between the hydrology, mass transport, and chemical kinetics of leaching. Such a change generally affects different streamlines in different ways, increasing the production rate of some while decreasing that of others. This suggests that the most appropriate level of analysis for a field problem involving two-dimensional fluid flow is the individual streamline.

The uranium leaching computer simulation developed at the Bureau of Mines Twin Cities Research Center (TCRC) divides the leachant flow pattern into discrete hydrologic components (individual streamlines) and then models the chemistry and mass transport for each of these components separately. The model then computes the productivity and effectiveness of the entire pattern by summing the results from these individual streamlines. This approach permits analysis of the contribution of each streamline to the effectiveness and efficiency of the entire pattern.[5]

The model is presently capable of predicting the impact on streamline productivity of various operator-controlled parameters such as well configuration, pumping schedules, oxidant injection rate, and duration of operation, as well as site-dependent aquifer and ore zone parameters including aquifer permeability, ore grade (differentiating between oxidized and reduced uranium), the presence of other minerals which compete with uranium for oxidant, and accessibility of in situ uranium to the leach solution.

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[1] Operations research analyst.
[2] Mathemetician.
[3] Geologist.
[4] Mathematician.
All authors are with the Twin Cities Research Center, Bureau of Mines, Minneapolis, Minn.

[5] This modeling procedure utilizes research and modeling work performed at the University of Texas at Austin. For a detailed description of that modeling work, see--
Bommer, P. M., and R. S. Schechter. Mathematical Modeling of In-Situ Uranium Leaching. Soc. Petrol. Eng. J., v. 19, no. 6, December 1979, p. 393.

In addition to a brief description of the chemical kinetic model, which has three constituent models, some examples of the graphic output will be presented and discussed.

MODEL CONFIGURATION

The kinetic geochemical model is composed of three component models. Two of these components, hydrology and mass transport, are computer-based models. The third, oxidation rate chemistry, is a laboratory model of the leaching process.

The overall model configuration is illustrated in figure 1. The hydrology and laboratory geochemical models must be run prior to the mass transport model, which, in addition to simulating the effects of convection and dispersion, performs the function of integrating hydrologic and geochemical output.

The hydrology model draws primarily on site-specific well, pumping, and aquifer characteristics for input, while the laboratory geochemical model involves site-specific ore material and leachant. Additional dispersion characteristics of the aquifer are input to the mass transport model, and the final product is a site-specific chemical kinetic simulation.

The predictive output from these three models includes the streamline flow pattern, the hydraulic head and fluid velocity throughout the aquifer, the concentrations of uranium and oxygen in the leach solution along each streamline, and the uranium recovered from individual streamlines. In addition, the uranium production from individual wells, and for the entire pattern, is computed.

Hydrology Model

The hydrology model assumes two-dimensional steady state flow in a homogeneous aquifer. Anisotropy characterized by discrete zones of differing permeabilities, either naturally occurring or induced by leaching, is permitted.

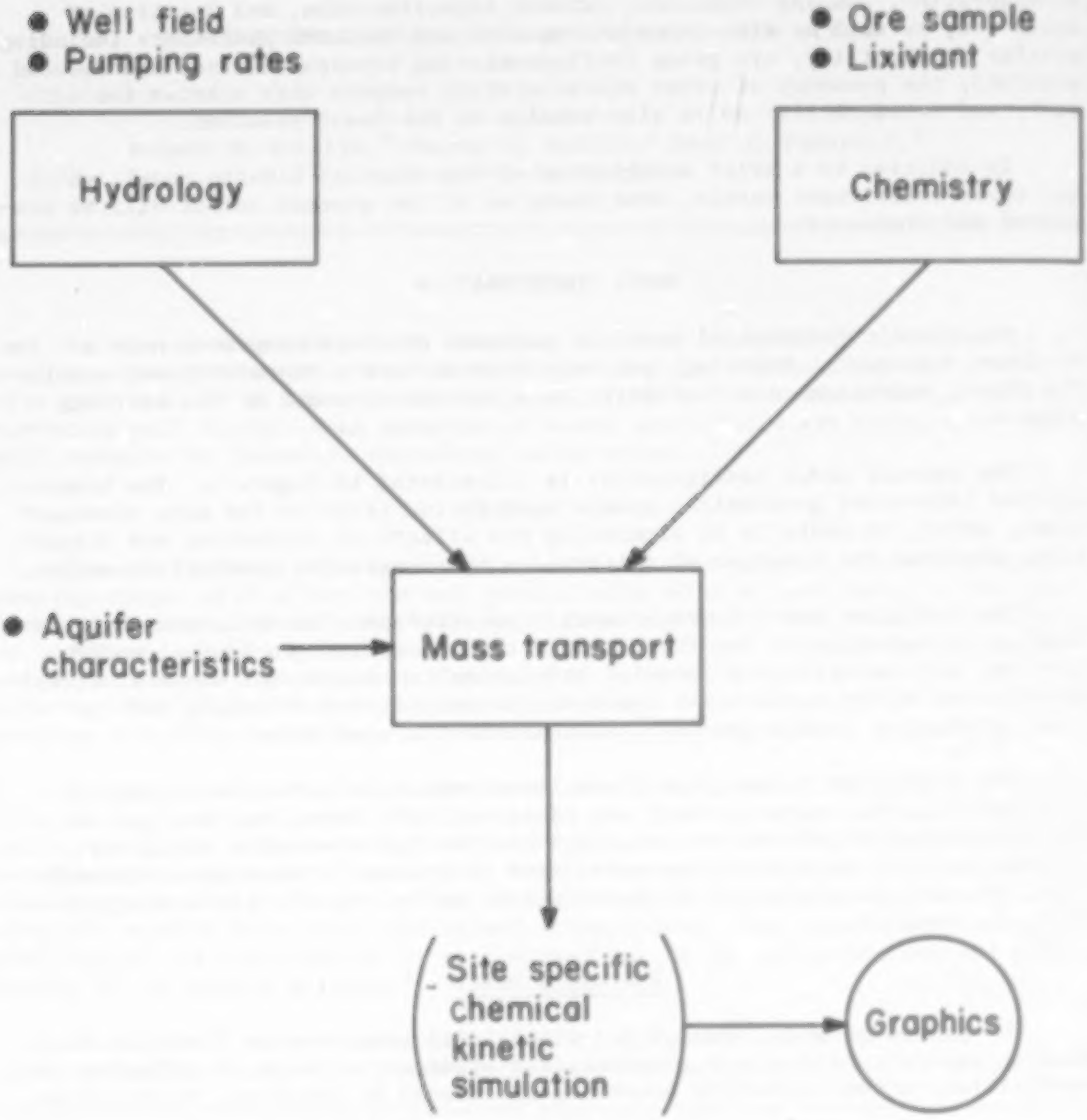
● Well field
● Pumping rates
Hydrology
● Ore sample
● Lixiviant
Chemistry
● Aquifer characteristics
Mass transport
Site specific chemical kinetic simulation
Graphics

FIGURE 1. - Model configuration.

The fundamental equation describing two-dimensional flow in a homogeneous isotropic aquifer without accretion is derived from Darcy's law and from the fluid conservation law and is given by Laplace's equation,

$$\frac{\partial^2 \phi}{\partial x^2} + \frac{\partial^2 \phi}{\partial y^2} = 0, \tag{1}$$

where

$$\phi = \phi(Z) \text{ for } Z = x + iy \tag{2}$$

and ϕ is specified on the boundary of the region, $Z = Z_c$, by

$$\phi(Z_c) = \phi_c. \tag{3}$$

The complex potential function $\Omega = \Omega(Z)$ is defined as an analytic function whose real and imaginary parts are, respectively, the potential function, $\phi = \phi(Z)$, and its harmonic conjugate, the streamline function, $\Psi = \Psi(Z)$.

With the definitions of ϕ and Ψ as given, the Cauchy-Remann conditions

$$\frac{\partial \phi}{\partial x} = \frac{\partial \Psi}{\partial y} \text{ and } \frac{\partial \Psi}{\partial x} = - \frac{\partial \phi}{\partial y} \tag{4}$$

insure that the real part of $\Omega(Z)$, $\phi(Z)$, will satisfy equation 1.

The hydrology output consists of a streamline flow net along with the fluid velocity and hydraulic head values along each streamline. The pattern resulting from a typical five-spot pattern and a five-spot with guard wells is pictured in figure 2.

Oxidation Rate Model

The chemistry model consists of a pressurized column leaching apparatus and involves a series of leaching experiments with different injection concentrations of dissolved oxygen. The intent is to determine an empirical relation between the oxygen concentration and the rates of uranium and mineral (pyrite) oxidation. This ore-specific relation is then incorporated into a chemical kinetic expression along with the simulated hydrologic output and applied to the field problem.

The geochemical predictions are based on the assumption that oxidation is the rate-controlling step in the uranium leaching process.

The laboratory apparatus involved in these experiments is shown in figure 3. The ore sample is contained in a cylindrical elastic membrane inside the stainless steel flow cell at the left of the picture. The confining pressure of overburden is simulated in the laboratory by pressurizing the annular space between the elastic membrane and the flow cell.

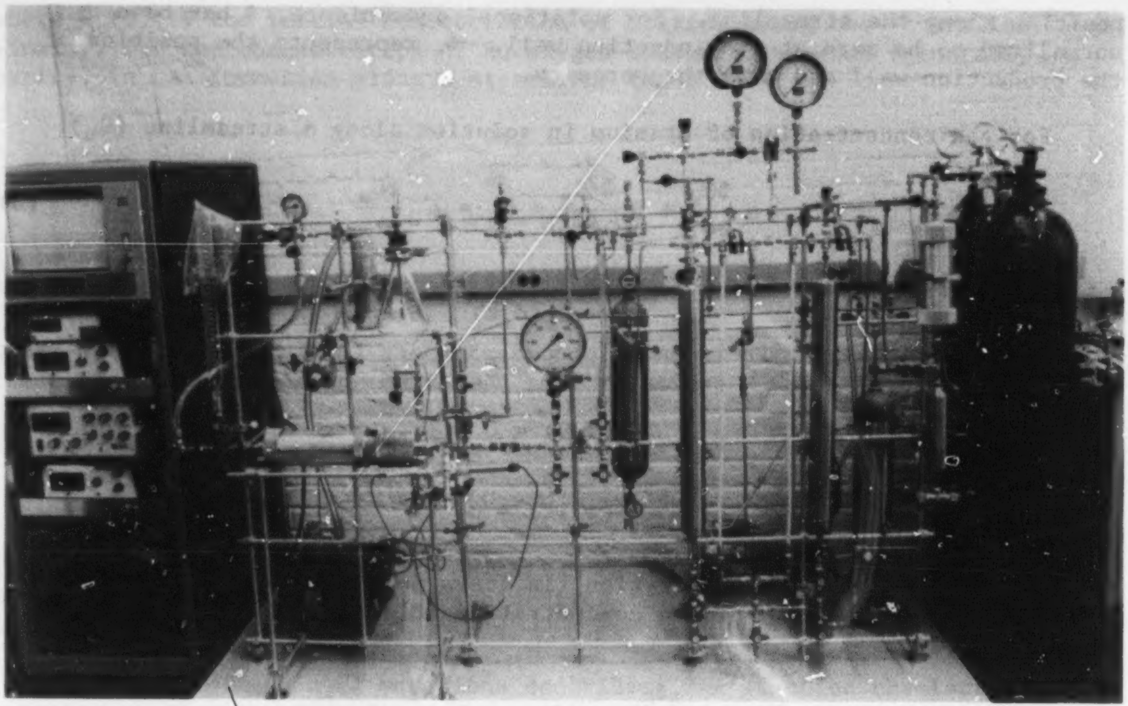

FIGURE 3. - Laboratory leaching apparatus.

Mass Transport Model

The mass transport model accounts for the flow and dispersion of leach solution through the aquifer. Only longitudinal dispersion is simulated; transverse dispersion (across streamlines) is neglected.

The mass transport model, of necessity, performs the additional function of integrating the hydrology output (flow lines, velocity, and hydraulic head along streamlines) and the laboratory chemistry results (uranium and pyrite oxidation rates) into a kinetic chemical expression. Thus, mass transport modeling is performed subsequent to the hydrology and laboratory chemistry simulations.

The integrated kinetic chemical model involves a numerical solution to a system of one-dimensional partial differential equations describing the change in uranium and oxygen concentration along streamlines as a result of convection, dispersion, and the chemical oxidation processes involved in leaching.

The basic, one-dimensional, convection-dispersion equations with reaction terms are presented below along with the appropriate boundary conditions. Note that the curvilinear coordinate system $\Omega = \Phi + i\Psi$, used in the hydrology model, is again employed to describe two-dimensional flow. Since Ψ is constant along each streamline, the mass transport equation for each streamline reduces to a one-dimensional flow problem with Φ as the variable representing position along the streamline. For notational convenience, Φ has been normalized to be zero at the injection well. Φ_n represents the position of the production well and ϕ is the porosity.

For the concentration of uranium in solution along a streamline (C_u),

$$\alpha \frac{\partial C_u}{\partial \Phi} + \beta \frac{\partial^2 C_u}{\partial \Phi^2} + R_u = \phi \frac{\partial C_u}{\partial t} \tag{5}$$

with

$$C_u = C_u(\Phi,t)$$

$$C_u(0,t) = \bar{C}_u$$

$$C_u(\Phi,0) = 0$$

$$\frac{\partial C_u}{\partial \Phi}(\Phi_n,t) = 0$$

where $\bar{C}_u$ is the uranium concentration (in grams per cubic centimeter) in the injected lixiviant. For the concentration of dissolved oxygen along a streamline (C_o),

$$\alpha \frac{\partial C_o}{\partial \Phi} + \beta \frac{\partial^2 C_o}{\partial \Phi^2} - (\delta R_o + \mu R_u) = \phi \frac{\partial C_o}{\partial t} \tag{6}$$

with

$$C_o = C_o(\Phi,t)$$

$$C_o(0,t) = \bar{C}_o$$

$$C_o(\Phi,0) = 0$$

$$\frac{\partial C_o}{\partial \Phi}(\Phi_n,t) = 0$$

where $\bar{C}_o$ is the oxygen concentration (in grams per cubic centimeter) in the injected lixiviant and δ and μ is a stoichiometric constants. For the uranium concentration in the deposit along a streamline (W_u),

$$\frac{\partial W_u}{\partial t} = -R_u \tag{7}$$

with

$$W_u = W_u(\Phi,t)$$

$$W_u(\Phi,0) = \bar{W}_u(\Phi)$$

where $\overline{W}_u(\Phi)$ is the initial concentration of uranium along the streamline. Finally, for the concentration of an oxygen-consuming mineral (generally pyrite) in the formation along a streamline (W_p),

$$\frac{\partial W_p}{\partial t} = -R_o \tag{8}$$

$$W_p = W_p(\Phi,t)$$

$$W_p(\Phi,0) = \overline{W}_p(\Phi)$$

where $\overline{W}_p(\Phi)$ is the initial concentration of mineral along the streamline. In these four expressions the oxidation rate relations R_u and R_o, given by

$$R_u = R_u\ (C_u,\ C_o,\ W_u,\ W_p)$$

$$R_o = (C_u,\ C_o,\ W_u,\ W_p),$$

are derived from the laboratory oxidation rate experiments. The longitudinal dispersion coefficients α and β are derived from fluid flow parameters output by the hydrology model.[6]

These one-dimensional equations describing the reaction chemistry and mass transport are solved for each streamline. The calculation of the uranium recovery for the two-dimensional well pattern is simply the total uranium produced by a radial pattern of streamlines around each injection well.

MODEL APPLICATIONS

The remainder of this paper presents the graphical results of using the model to simulate a hypothetical five-spot operation. These graphical descriptions permit the user to readily interpret the results of a simulation and are critical for gaining insight into the attributes of different leaching strategies. The model developed at TCRC incorporates a number of computer programs capable of graphically representing concentration profiles and histories for uranium or oxygen, and recovery curves for streamlines, individual wells, and the entire pattern.

Figure 4 shows a series of concentration profiles generated by the model which depict the concentration of uranium in solution along the length of the laboratory flow cell shown in figure 3. One pore volume of leachant is pumped through the cell, and eight concentration profiles are developed at equal time intervals. Such a simulation of the laboratory leaching process "tunes," or calibrates, the model prior to applying it (with its site-specific parameter values and laboratory-derived oxidation rate expressions) to the field problem. Discrepancies between the computer-simulated profiles of figure 4 (the endpoints only) and the measured uranium concentration in solution discharged

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[6] Bommer, P. M. A Streamline-Concentration Balance Model for In Situ Uranium Leaching and Site Restoration. Ph.D. Dissertation, Univ. Texas, Austin, Tex., August 1979, 263 pp.

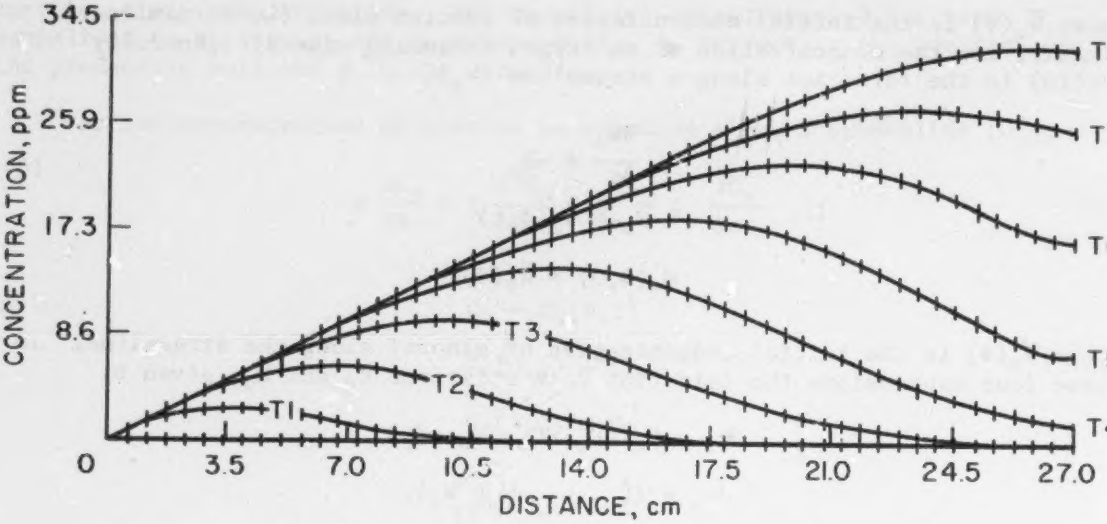

FIGURE 4. - Laboratory flow cell uranium concentration profiles.

from the laboratory flow cell can be minimized by manipulating certain program inputs. Very often, measured parameter values are only approximate. For example, permeability inputs are frequently the result of an averaging process.

Besides acquainting the user with the dynamics of streamline chemical kinetics, the subsequent computer graphics point out the hydrologic and geochemical differences between interior and exterior streamlines of a five-spot pattern. Further, these examples demonstrate the advantage of analyzing streamlines individually, and display the simulator's usefulness as a predictive analytic tool.

Figure 5 shows a short concentration history (8 days) of two different streamlines on a typical five-spot pattern of wells. In each case, the streamline under consideration is indicated by a dashed line on the five-spot pattern at the upper right of the illustration. Each streamline is considered to be the center of a region of the flow pattern that is commonly referred to as a streamtube. This short history is specifically intended to show the difference in front breakthrough times at the recovery well--approximately 15 hours for figure 5A versus 30 hours for figure 5B.

Each concentration history depicted in figure 5 is the uranium concentration at the last point on the streamline (that is, the center production well) plotted as a function of time. Thus, these histories reflect the contribution made by each of the two streamtubes to the total uranium recovered from the well. The obvious difference in the magnitudes of the two curves illustrates the variability of performance among streamlines of a five-spot well pattern. The dissimilar paths of these two streamlines lead to the conjecture that the differences shown in figure 5 are a result of (1) the greater areal sweep of

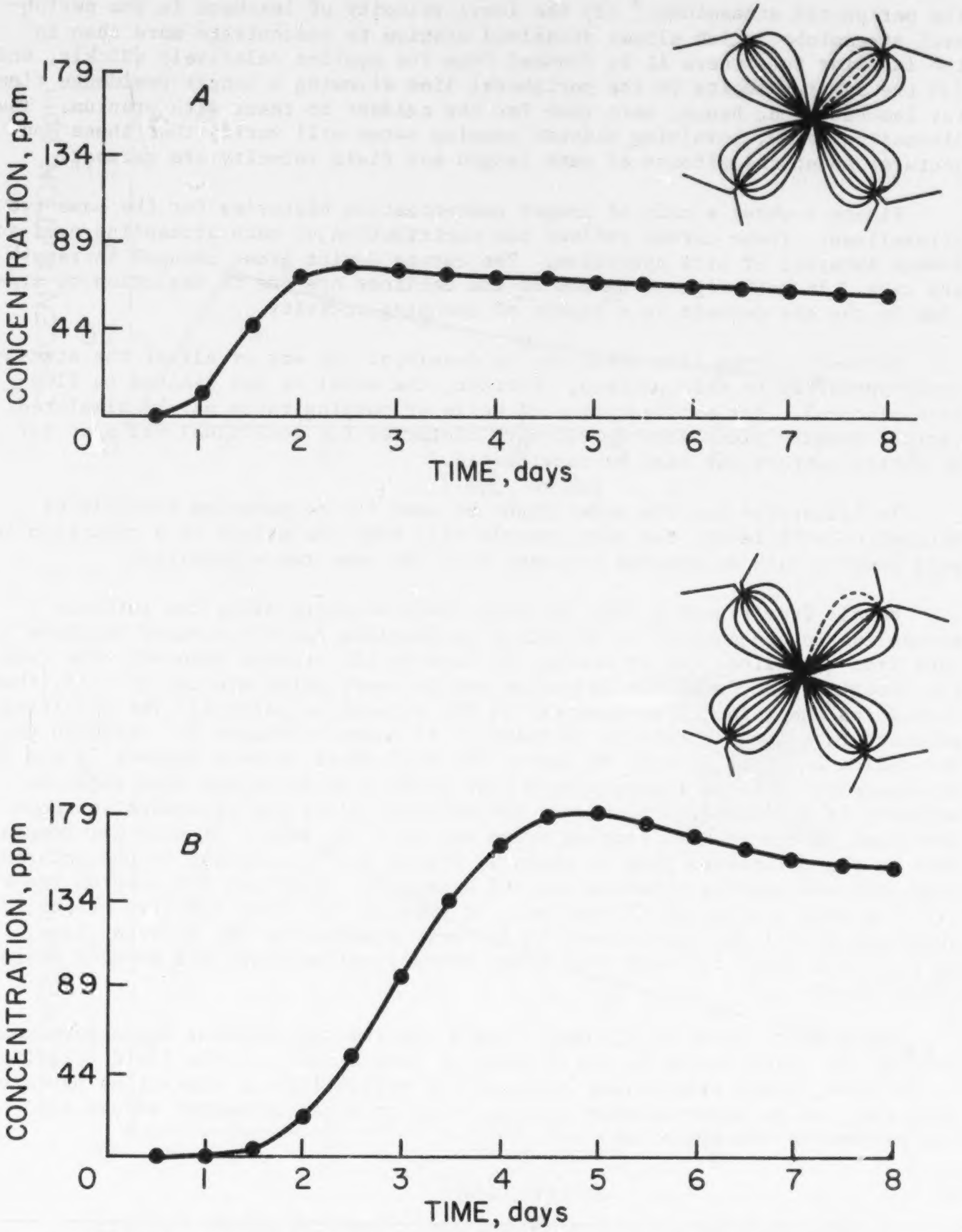

FIGURE 5. - Eight-day uranium concentration histories. *A*, Interior streamline; *B*, peripheral streamline.

the peripheral streamtube,[7] (2) the lower velocity of leachant in the peripheral streamtube, which allows dissolved uranium to concentrate more than in the interior tube where it is flushed from the aquifer relatively quickly, and (3) the lower velocity in the peripheral line allowing a longer residence time for leachant and, hence, more time for the oxidant to react with uranium. The discussion below, involving reduced pumping rates will verify that these conjectures about the effects of path length and fluid velocity are correct.

Figure 6 shows a pair of longer concentration histories for the same two streamlines. These curves reflect the contribution of each streamline over an 8-week interval of site operation. The curves depict gross changes in recovery rate. In both figures 6A and 6B the declines are due to depletion of uranium in the ore deposit as a result of leaching activity.

Recovery curves like these can be developed for any or all of the streamlines appearing in this pattern. Further, the model is not limited to five-spot patterns. Any configuration of wells or pumping rates may be simulated. Various summary plots showing recovery histories for individual wells or for an entire pattern can also be constructed.

To illustrate how the model might be used for comparative analysis of streamline efficiency, the next example will show the effect of a reduction in well pumping rate on uranium recovery from the same two streamlines.

Recall from figure 5 that the high fluid velocity along the interior streamline is conjectured to be partly responsible for the reduced recovery rate from this line. In an attempt to improve the uranium recovery rate from the interior line, solution injection and recovery rates are cut in half, thus decreasing the velocity of leachant in the streamline pattern. The resulting recovery rate for the interior streamline is shown in figure 7. Although the simulation interval is only 80 hours, the difference between figures 7A and 7B is apparent. Uranium recovery from this streamline increases when solution velocity is decreased. Of course, the velocity along the peripheral streamline also decreases when pumping rates are cut; the impact of this new pumping rate on the peripheral line is shown in figure 8. In contrast to the interior line, the new pumping schedule has had a negative impact on the uranium recovery from this peripheral streamline. It appears that this negative effect far outweighs the slight improvement in recovery achieved by the interior line. As a result, total recovery from these two streamlines over the 80-hour period has declined.

The results shown in figures 7 and 8 confirm the previous conjecture concerning the roles played by the streamline path length and the fluid velocity. Furthermore, these simulations evidence the variability in streamline performance that can be expected when pumping rates or other parameter values are manipulated by the operator.

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[7] The amount of uranium accessible to the leachant being proportional to the area swept.

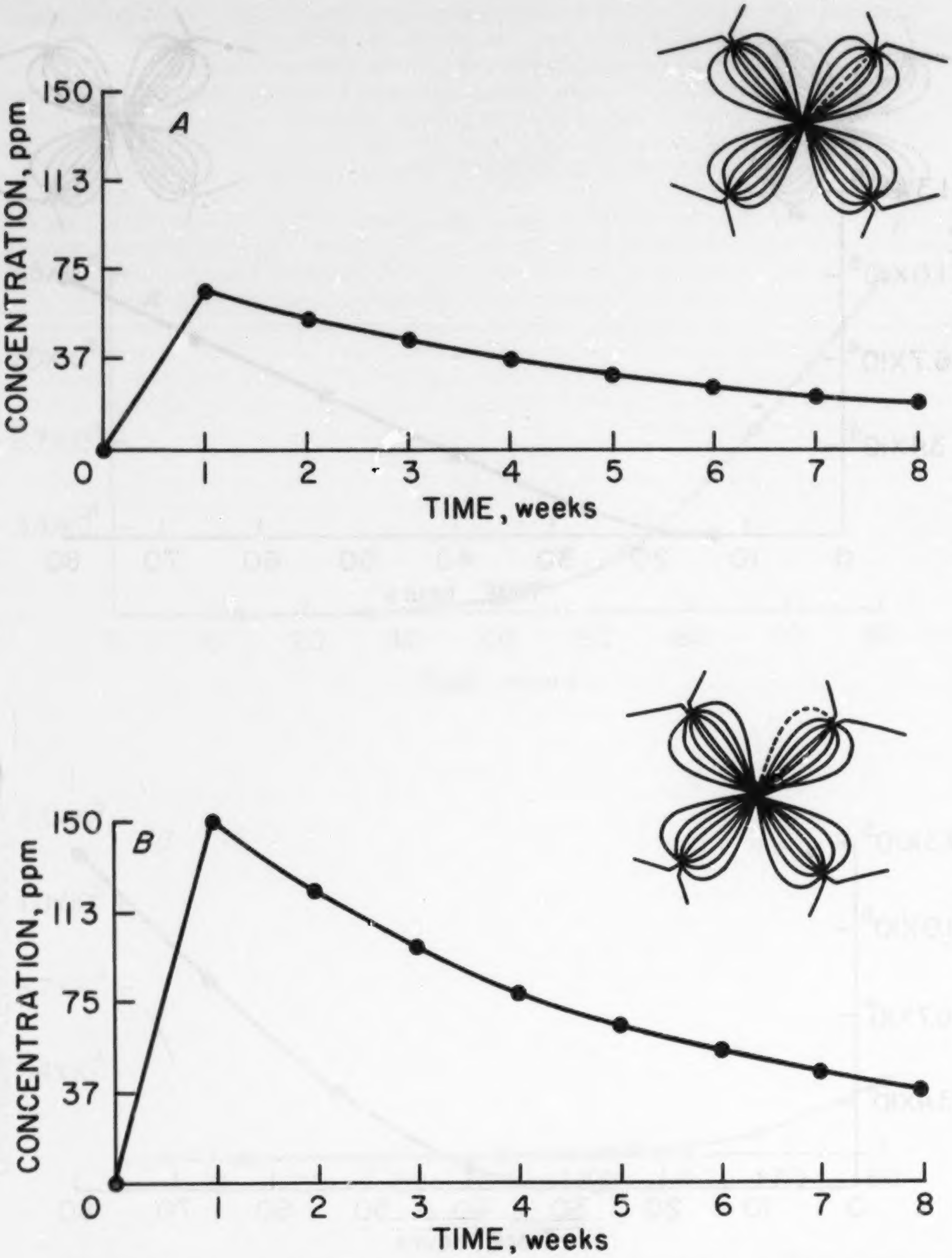
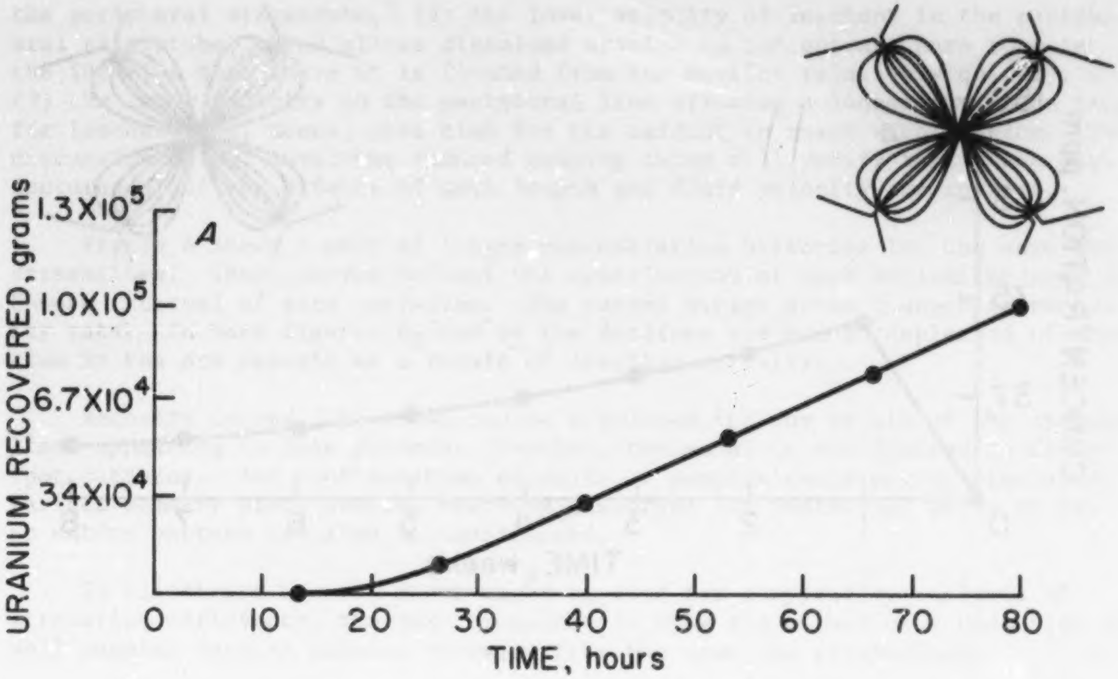


FIGURE 6. - Eight-week uranium concentration histories. *A*, Interior streamline; *B*, peripheral streamline.



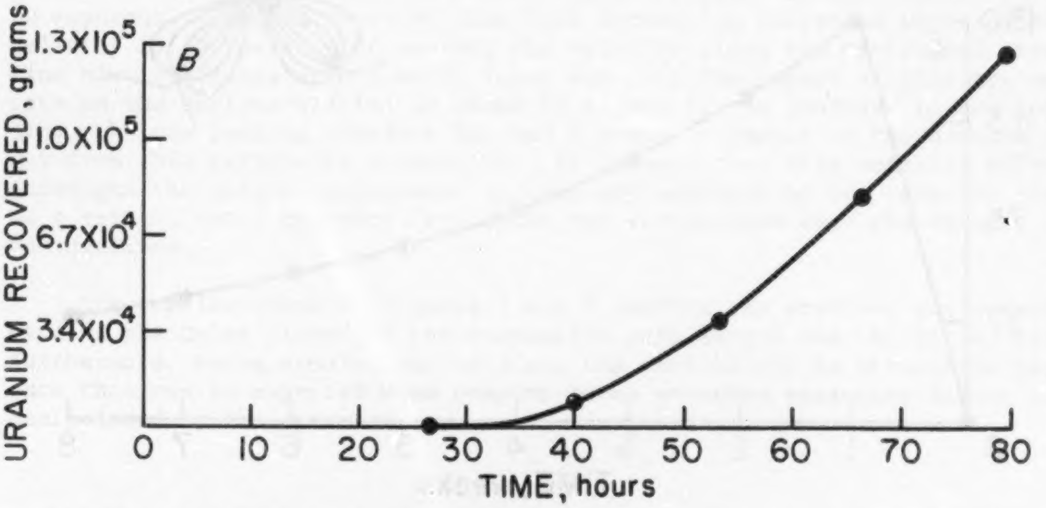


FIGURE 7. - Uranium production graphs for interior streamline. ***A*****, Full injection rate (15 gpm);** ***B*****, halved injection rate (7.5 gpm).**

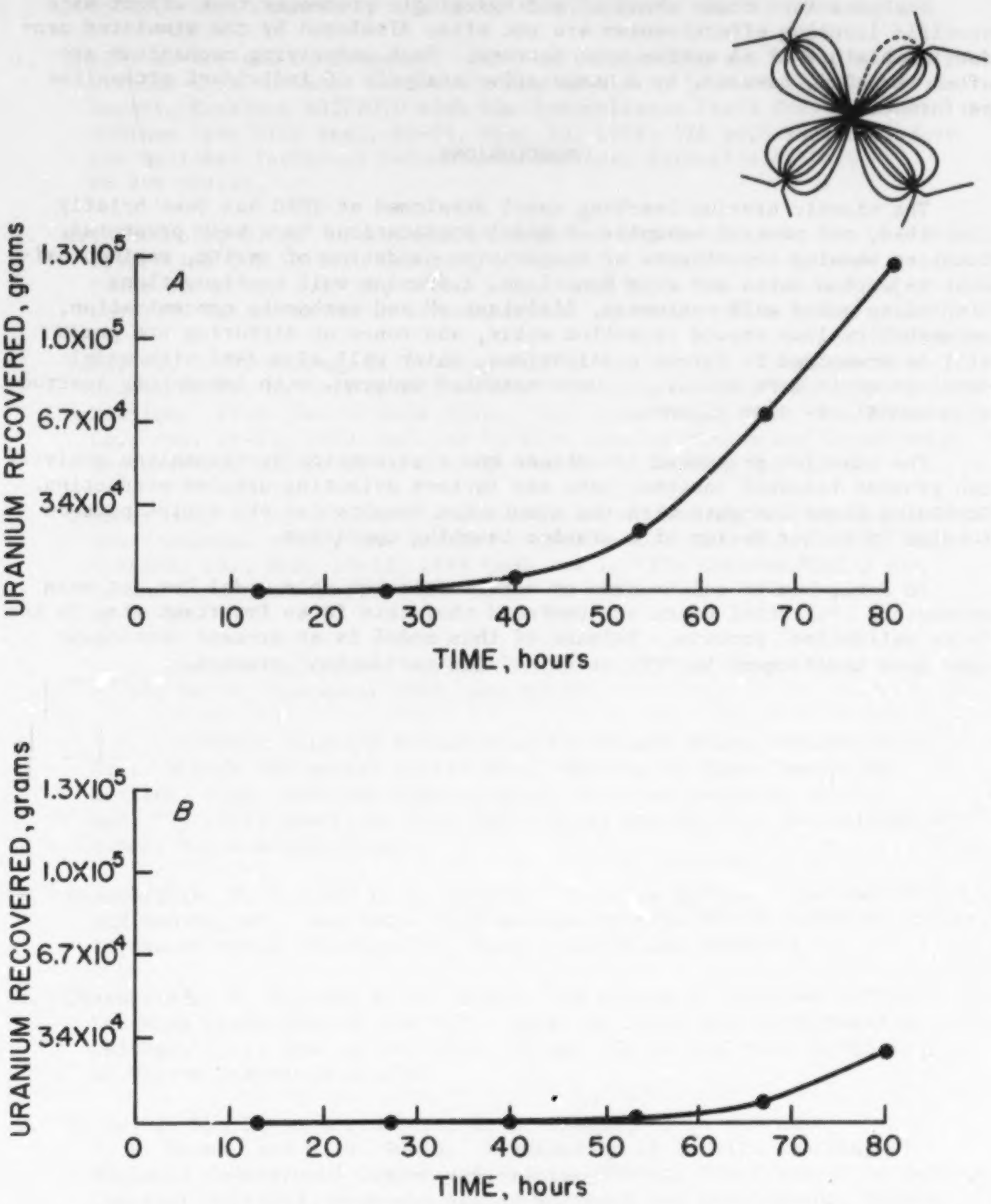


FIGURE 8. - Uranium production graphs for peripheral streamline. *A*, Full injection rate (15 gpm); *B*, halved injection rate (7.5 gpm).

Insights into those chemical and hydrologic processes that affect site specific leaching effectiveness are not often disclosed by the simulated production history of an entire well pattern. Such underlying mechanisms are often revealed, however, by a comparative analysis of individual streamline performance.

CONCLUSIONS

The kinetic uranium leaching model developed at TCRC has been briefly described, and several examples of model applications have been presented. Examples showing the effects of competitive oxidation of pyrite, various oxidant injection rates and rate functions, different well configurations (including guard well patterns), lixiviant pH and carbonate concentration, permeability loss around injection wells, and zones of differing ore grade will be presented in future publications, which will also deal with model development in more detail. A user-oriented program, with laboratory instruction manual, is also planned.

The examples presented illustrate how a streamline-by-streamline analysis can provide valuable insights into the factors affecting uranium production. Combining these insights with the simulation results for the entire pattern results in better design of a uranium leaching operation.

To date, onsite calibration or verification of this model has not been attempted. Potential users are reminded that this is an important step in the "site validation" process. Release of this model is at present contingent upon some involvement by TCRC in this "site validation" process.

APPENDIX.--BIBLIOGRAPHY OF BUREAU OF MINES IN SITU MINING PUBLICATIONS

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